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CANMET
Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

PROCEEDINGS OF THE MEETING OF THE CANADIAN URANIUM PRODUCERS' METALLURGICAL COMMITTEE HELD IN MISSISSAUGA, MAY 19-20, 1983

G.M. Ritcey
Extractive Metallurgy Laboratory

April 1984

Project - 4.4.1.0.01
Conventional Uranium Extraction Technology

This document is an unedited interim report prepared primarily for discussion and internal reporting purposes. It does not represent a final expression of the opinion of the Canada Centre for Mineral and Energy Technology (CANMET)

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ENERGY RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES
DIVISION REPORT ERP/MSL 84-53(TR)

MICROMEDIA
JUL 28 1984

PREFACE

This publication has been prepared to provide a permanent record of the Proceedings of the 1983 Annual Meeting of the Canadian Uranium Producers' Metallurgical Committee which was held in Toronto, Ontario. The author's original unedited technical presentations together with the discussions have been incorporated into this report.

The next annual meeting of the Committee is scheduled for May 1984 and will be held in Blind River, Ontario.

ACKNOWLEDGEMENTS

The participation of those who attended is to be commended and is the main factor that demonstrates the usefulness of the meeting. The generous help and hospitality provided by Ontario Research Foundation and the tours to ORF, INCO, Witteck Development and Gulf Research certainly added to the success of the meeting. Acknowledgement is also made to Mrs. L. Paquette, CANMET, who assisted in the preparation of this record of the meeting.

G.M. Ritcey
Secretary, CUPMC

BUSINESS SESSION

A short business meeting concluded the two-day meeting. The bank balance was stated as $266.90 as of April 30, 1983.

The next meeting of the CUPMC was set for May 1984 in Blind River, Ontario, with Eldorado Nuclear the host, and Don Francis as Chairman.

Ontario Research Foundation was thanked for providing the arrangements and hospitality of the meeting and the tours of ORF. Thanks were expressed to INCO, Witteck Development and Gulf Research for the tours of their research laboratories.

CANADIAN URANIUM PRODUCERS' METALLURGICAL COMMITTEE
1983 ANNUAL MEETING
MAY 19-20, 1983

SHERIDAN PARK CONFERENCE CENTER, MISSISSAUGA, ONTARIO
(SEE ATTACHED MAP)

CHAIRMAN: V. LAKSHMANAN, ONTARIO RESEARCH FOUNDATION

PROGRAM

THURSDAY, MAY 19, 1983

0930-1000	Coffee
1000-1010	Opening Remarks
1010-1050	Background Radiation in Canada - R. Grasty, Geological Survey of Canada
1050-1130	Canadian Uranium Deposits and Resources - V. Ruzicka, Geological Survey of Canada
1130-1210	Migration of Contaminants from Underwater Buried by Pyritic Uranium Tailings — Laboratory Investigations of Radium-226 Transport Flux - N. Davé, CANMET
1210-1245	Report of the Analytical Subcommittee - L. Dalton, CANMET
1245-1345	Lunch at Conference Centre
1345-1425	Uranium Waste Management in Canada - L. Melis, Consultant
1425-1505	Thickened Discharge Disposal of Tailings, E. Rubinsky, University of Toronto
1505-1530	Coffee
1530-1620	Decommissioning of the Beaverlodge Mine-Mill Operations and Reclamation of the Site - M. Filion, Eldorado Nuclear Ltd.
1620-1710	Closing Down the Beaverlodge Mill - G. Feasby, Consultant
1730-1830	Cash Bar
1830	Dinner at Conference Center

FRIDAY, MAY 20, 1983

0830-0900	Coffee
0900-0920	Activities of the NEA/IAEA Working Group on Uranium Extraction - M. Campbell, CANMET
0920-1000	Reduction of Tailings Radium Levels Using Modified Milling Processes - M. Baird, McMaster University
1000-1040	Future Process Possibilities - G. Ritcey, CANMET
1040-1110	Coffee
1110-1150	Long Term Plans for Uranium Tailings Research - V. Haw, National Uranium Tailings Program
1150-1230	Eldorado's Blind River Refinery - D. Francis and J. Schisler, Eldorado Nuclear Ltd.
1230-1240	Business Meeting
1245	Lunch

PM - Several organizations have kindly offered tours of their facilities. Because of the time available only one site can be visited by any one person. Therefore indicate on the attached form your tour preference.

CUPMC ATTENDANCE, MAY 19-20, 1983

NAME	AFFILIATION
N. Aspin	Canadian Nuclear Association, Toronto
M. Baird	McMaster University, Hamilton
E. Barnes	Rio Algom Mines Ltd., Elliot Lake
L. Bates	Kilborn Engineering, Toronto
R. Beyette	Eldorado Resources Ltd., Blind River
C. Booth	Ontario Research Foundation, Mississauga
H. Branstatter	Ontario Research Foundation, Mississauga
M. Campbell	CANMET, Ottawa
R. Chapman	Kidd Creek Mines, Timmins
N. Chiu	Monenco Consultants, Calgary
B. Chomyn	AECL, Ottawa
L. Dalton	CANMET, Ottawa
N. Davé	CANMET, Elliot Lake
T. Edwards	Willowdale
C. Farrow	Bechtel Canada Ltd., Toronto
G. Feasby	NUTP, Ottawa
M. Filion	Eldorado Resources Ltd., Ottawa
P. Fournier	AMOK Ltd., Saskatchewan
D. Francis	Eldorado Resources Ltd., Blind River
J. Goode	Kilborn Ltd., Toronto
W. Gow	CANMET, Ottawa
R. Grasty	Geological Survey of Canada, Ottawa
D. Green	Dorr-Oliver Canada, Orillia
V. Haw	NUTP, Ottawa
A. Hayden	A.H. Ross & Assoc., Toronto
K. Hester	K.D. Hester & Assoc., Oakville
L. Heymann	Agnew Lake Mines, Espanola
I. Itzkovitch	Eldorado Resources Ltd., Ottawa
E. Joe	CANMET, Ottawa
R. John	NUTP, Ottawa
I. Joyce	Ontario Research Foundation, Mississauga
F. Kelly	CANMET, Ottawa
E. Krause	INCO Ltd., Mississauga
R. Knapp	Senes Consultants Ltd., Willowdale
V. Lakshmanan	Ontario Research Foundation, Mississauga
T. Lim	CANMET, Elliot Lake
M. Lopez	Interox America, USA
L. Luckevich	Ontario Research Foundation, Mississauga
D. Lush	IEC Beak Consultants Ltd., Mississauga
L. McFarlane	Madawaska Mines Ltd., Bancroft
J. Meinbardis	Ontario Research Foundation, Mississauga
L. Melis	Saskatoon
F. Melvanin	H.G. Engineering Ltd., Don Mills
W. Milde	Eldorado Resources Ltd., Saskatoon
R. Nice	Pancontinental Mining Ltd., Toronto
A. Oliver	Eldorado Resources Ltd., Ottawa
G. Ritcey	CANMET, Ottawa
J. Roberts	MacLaren Plansearch, Toronto
J. Robertson	Ministry of Natural Resources, Toronto
E. Rubinsky	E.I. Rubinsky Assoc. Ltd., Toronto
D. Rollwagen	Witteck Dev. Inc., Mississauga

NAME	AFFILIATION
V. Ruzicka	Geological Survey Of Canada, Ottawa
J. Schisler	Eldorado Resources Ltd., Blind River
R. Swider	Toronto
E. Van Huyssteen	Ontario Research Foundation, Mississauga
E. Ward	Toronto
R. Whillans	Uranium & Nuclear Energy, Ottawa
F. Widmar	Witteck Dev. Inc., Mississauga

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NATURAL BACKGROUND RADIATION IN CANADA

R.L. Grasty, J.M. Carson, B.W. Charbonneau
and P.B. Holman

Prepared for a Geological Survey of Canada Paper

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NATURAL BACKGROUND RADIATION IN CANADA

R.L. Grasty, J.M. Carson, B.W. Charbonneau
and P.B. Holman

ABSTRACT

Published airborne gamma-ray survey data from 33 areas of Canada were used to compile information on the average ground level exposure from natural radiation. The exposures at ground level were calculated from the surface concentrations of potassium, uranium and thorium and were confirmed over a calibration strip by measurements with an ionization chamber.

The highest levels of radioactivity were found in northern Canada and were generally related to granitic rocks. The lowest areas of radioactivity were associated with the Athabasca sandstone.

Summer outdoor exposure rates from potassium, uranium and thorium for the 33 areas had a population weighted average of 3.7 +/- 2.3 $\mu R\ h^{-1}$, of which 48 percent originated from potassium, 43 percent from the thorium series and 9 percent from the uranium series. This low

level of radioactivity, compared to world-wide data, can be explained by a simple model of crustal evolution in which the radioactivity of the crust decreases with depth. When seasonal variations of soil moisture and snow cover are considered, the annual population weighted average outdoor exposure rate from potassium, uranium and thorium decreases to 2.9 +/- 1.8 μR h^{-1} corresponding to an annual outdoor dose-equivalent of 18 +/- 11 mrem (180 +/- 110 μSv).

Additional components of outdoor annual dose-equivalent arise from cosmic radiation (32 +/- 3 mrem - 320 + 30 μSv) and the internal radioactivity of the body (19 mrem - 190 μSv) which give a total outdoor annual value of 69 +/- 11 mrem (690 +/- 110 μSv). Based on a comparison of indoor and outdoor values from world-wide published data, the average annual Canadian whole-body dose-equivalent from all sources of natural radiation is estimated to be 75 +/- 17 mrem (750 +/- 170 μSv).

INTRODUCTION

The major source of radiation exposure to man arises from the natural environment. This natural radiation is therefore frequently used as a standard for comparing additional sources of man-made radiation such as those produced by nuclear power generation, waste disposal, atomic weapons fallout and medical sources of x-rays. In order to assess the significance of these additional sources of man-made radiation the levels of the natural background radiation and its variation must be known.

Natural radiation exposure originates from both internal and external sources. Internal sources comprise naturally occurring radioactive elements such as potassium-40 and the gas radon-222 which are taken into the body. External sources of radiation originate from cosmic rays and natural radioactive elements, principally potassium-40 and decay products in the uranium and thorium decay series occurring in the ground, in building material and in the air.

The decay of natural radio-nuclides produces alpha-particles, beta-particles and gamma-radiation. Almost the entire energy of alpha and beta particles is absorbed by a few millimeters of human tissue. Consequently, apart from the surface of the skin the radiation received by the body from external sources of radioactivity is due almost entirely to gamma radiation. This external radiation can vary considerably for each person depending on such things as his geological environment, elevation above sea level and type of living accommodation.

In the case of internal sources of radioactivity, the lung and associated organs receive a much greater radiation dose than the rest of the body because of alpha and beta particles emitted by radon-222 and its decay products. This paper considers the effects of gamma radiation and cosmic radiation on the whole body and is not concerned with alpha and beta particles which effect specific organs.

Measurements of natural background radiation have been performed in many parts of the world using a variety of different techniques. These techniques have involved the use of ionization chambers as well as portable and airborne scintillometers. Laboratory analyses of the radioactive elements in soil samples have also been used to estimate the average radiation exposure in several different countries. In using these techniques problems frequently arise. The measurements of Herbst (1964) in Switzerland were made with an ionization chamber at a time when the contribution from atomic weapons fall-out was as high as 30 percent of the total radiation from the ground

(Oakley, 1972). The results of Hultqvist (1956) in Sweden and Ohlsen (1969) in East Germany also include some contribution from weapons fall-out. The results of Ohlsen and Herbst as reported by the United Nations Scientific Committee on the Effects of Atomic Radiation (UNSCEAR, 1977) show the highest radiation levels reported for any country in the world. Since the radiation from fall-out has now decayed to negligible levels compared to those from natural radioactivity, these figures are no longer applicable.

Because of the variability of the ground radioactivity from place to place, with any form of ground survey it is difficult to carry out a representative sampling on a country wide basis. This is particularly true in large countries such as Canada or the United States. In the U.S. estimates of natural radiation levels have been made by Oakley (1972) and the National Council on Radiation Protection and Measurements (NCRP, 1975). They made use of nation wide airborne surveys of radioactivity in the vicinity of nuclear facilities. These Aerial Radiological Measurements Surveys (ARMS) were conducted by the U.S. Geological Survey and E.G. & G. Inc. for the U.S. Atomic Energy Commission between 1958 and 1963 and covered 25 areas representing approximately 30 percent of the population. Because of the large area covered, surveys of this nature provide a far superior method of estimating radiation levels than do ground surveys with a limited number of measurements. The ARMS surveys however were scintillometer surveys and could not distinguish between atomic weapons fallout and the radioactivity from potassium and the uranium and thorium decay series. In the period 1961 to 1963 radiation levels from weapons fall-out was 50 percent or more of the natural terrestrial values (Oakley, 1972). The accuracy of the results from the individual ARMS surveys is therefore controlled by the accuracy of the estimation of the fall-out during the time the survey was carried out. In addition no systematic calibration procedure was adopted to relate the airborne radioactivity measurements to ground level values. For the more recent airborne gamma ray spectrometer surveys operating under the U.S. National Uranium Reconnaissance Program (NURE), considerable attention has been paid to the problems of calibration. This involved the construction of large radioactive concrete calibration pads (Ward, 1978) and the selection of an airborne test range with accurately known concentrations of potassium, uranium and thorium (Geodata, 1977).

In Canada no program has specifically been developed for estimating the radiation levels experienced by the population from natural sources. Measurements of radiation exposure at a few selected sites are reported on a routine basis by the Radiation Protection Bureau of the Department of Health and Welfare through their quarterly reports. These measurements are made with thermo-luminescent dosimeters (TLDs) at sites selected normally for their proximity to a government facility. Due to the limited number of sites involved, the TLD measurements are unsuitable for estimating average radiation levels.

In 1967 an airborne gamma ray spectrometer survey program commenced at the Geological Survey of Canada (GSC). This program was designed specifically for estimating ground concentrations of potassium, uranium and thorium to aid in geological mapping and uranium exploration. At the beginning of the program considerable effort was spent on the problems of calibration through the construction of large radioactive calibration pads at Uplands Airport, Ottawa (Grasty and Darnley, 1971) and the development of an airborne test range. This 7 km test range was systematically sampled to determine its potassium, uranium and thorium concentration (Grasty and Charbonneau, 1974).

Airborne gamma ray surveys have now been flown in many parts of Canada and are published as Geological Survey of Canada maps or open file reports. This published radioactivity data is presented in the form of contour maps of the concentrations of the three radioelements, as well as individual profiles along aircraft flight lines. Total radioactivity data which represents all gamma radiation above an energy of 0.41 MeV is also presented as well as the ratios of the three radioelements for use in mineral exploration. One such profile from a survey carried out in Nova Scotia is shown in Figure 1.

This paper shows how such airborne data can be used to estimate the outdoor radiation exposure from potassium, uranium and thorium at the surface of the ground. The data are then used to derive an average outdoor radiation exposure for each area surveyed and for that part of Canada covered by these surveys. The average annual outdoor radiation dose to the Canadian population was then computed. In this computation seasonal soil moisture fluctuations, the shielding effect of snow, vegetation and the human body itself were considered as well as additional components of natural radiation from cosmic radiation, atmospheric radon and the internal radioactivity of the body.

TERMINOLOGY AND DOSE RELATIONSHIPS

Units of radioactivity can be confusing even to the practicing health physicist. In this section we attempt to explain the various units used in this report in a straightforward manner which we hope will be clear to the average scientific reader who is not a specialist in the field of radiation.

For almost all fields of science a unit of a physical measurement such as temperature, density etc. uniquely defines a particular property of a material. An ionizing radiation field, however, cannot in general be defined uniquely since it can consist of radiation with a complete range of energies and angular distributions. One way of comparing radiation fields is by means of an ionization chamber which measures the quantity of electrical charge released in a gas through absorption of the radiation. This type of measurement is most useful for the health physicist since it may be related to the physical damage that will occur in living cells.

The radiation intensity at a given place is termed its 'Exposure' and is measured by its ability to produce ionization at that place. The unit of exposure is the roentgen, R. One roentgen is defined as the quantity of X or gamma radiation that produces one electrostatic unit of charge of either sign in 1 ml of standard air. In this report all the original data has been compiled and presented in these units of exposure.

In 1956 a unit of radiation was adopted which applied to any form of ionizing radiation. This unit of 'absorbed energy' or 'dose' is the energy imparted by ionizing radiation to 1 gram of any material, at the particular point of interest. The unit of absorbed dose is the 'rad' (radiation absorbed dose) which is the deposition of an energy of 100 ergs per gram. In expressing the absorbed dose, the particular absorbing material under consideration must always be given.

Different types of radiation cause slightly different effects in biological tissues. For this reason, in comparing the effects of radiation on living systems, another derived unit, the roentgen equivalent man or rem is used. The dose in rems is the product of the dose in rads and a factor called the quality factor which depends on the relative biological effectiveness (RBE) of the radiation considered. This unit of dose is commonly called dose-equivalent or D.E. The rem is the dose from any radiation that produces biological effects in man equivalent to one rad of x-rays. Therefore

$$\text{D.E. (rems)} = \text{RBE} \times \text{rads} \qquad (1)$$

X-rays and gamma rays which are principally of concern in this report have an RBE value of 1.

For the health physicist who is concerned with the radiation dose absorbed by the body, the final results of our analyses are presented in units of rems. It is therefore necessary to explain in detail how the original exposure rate data was converted to units of rems. The conversion factors and procedures are those followed by UNSCEAR (1977). The air absorbed dose rate, D_a, can be directly related to the exposure rate, E, by the expression

$$D_a = aE \qquad \text{.................................} \qquad (2)$$

where a has the value 0.869 rad R^{-1} .

If some material, other than air is placed in a radiation field, the energy absorbed will not, in general, be the same as for air because the absorption coefficients will differ. In human tissue there is a higher proportion of electrons per gram than in air because of the presence of hydrogen. Since gamma rays lose much of their energy through Compton scattering with electrons, the mass absorption coefficient of tissue is higher than it is for air, the ratio of the two absorption coefficients being 1.10.

In assessing the absorbed dose for any particular organ of the body, the shielding effect of the body must be considered. The absorbed dose, D_o for any organ in the body is then given by

$$D_o = 1.10 \times g \times D_a \text{ .... } (3)$$

where g is called the geometry factor. For most organs of interest, the values of g are nearly equal (UNSCEAR, 1977). In this report we use a value of 0.75 as calculated by Bennett (1970).

Combining equations (1), (2) and (3) we find that for gamma radiation with an RBE of unity the dose-equivalent (D.E.) is related to the exposure E by the expression

$$D.E. = 0.869 \times 1.10 \times 0.75 \times E \text{ .... } (4)$$

This is the basic relationship used to convert the original exposure rate data to dose-equivalent. This equation shows that apart from the geometry factor g (0.75), the dose-equivalent, in rems, is very nearly equal to the exposure rate in roentgens.

In recent years it is becoming increasingly common to find quantities used in radiation protection being expressed in the new international system of units. These S.I. units are the gray (Gy) and the sievert (Sv).

The gray is the unit of absorbed dose corresponding to the rad and is the energy imparted by ionizing radiation to material corresponding to one joule per kilogram. The relation between the gray and the rad is:-

$$1\ \mathrm{Gy} = 100\ \mathrm{rad} = 1\ \mathrm{J\ kg^{-1}}$$

The sievert is the new unit for dose-equivalent coresponding to the rem, the relation being given by

$$1\ \mathrm{Sv} = 100\ \mathrm{rem}$$

In order to compare our results with published data presented in the old units, the practice in this report will be to quote values in the old units immediately followed by the new units in brackets.

THE AIRBORNE SYSTEM

Prior to 1978 the volume of the airborne gamma ray detectors flown by the Geological Survey of Canada was approximately 50 l of sodium iodide (thallium activated) distributed in twelve 22.96 x 10.2 cm (9 x 4 inch) crystals. The detector array was maintained at 38 degrees C (100 degrees F) in thermally insulated boxes to minimize spectral drift. Pulses from the detectors were fed into a 128 channel analyser from which four energy windows could be selected and the pulses accumulated in four scalers.

Figure 2 shows a typical gamma-ray spectrum recorded at the normal survey altitude of 120 m. Gamma-ray peaks at 2.62 MeV, 1.76 MeV and 1.46 MeV representing thallium-208 in the thorium decay series, bismuth-214 in the uranium decay series and potassium-40 respectively can be readily distinguished. These particular gamma rays have generally been accepted as most suitable for the measurement of potassium, uranium and thorium because they are relatively abundant and being high in energy are not appreciably absorbed in the air. The energy windows used to monitor these particular gamma rays are shown in Table 1. The total count window reflects general lithological variations and is therefore useful in geological mapping. The accumulated counts in these windows were recorded digitally on magnetic tape together with details of aircraft altitude, navigational information and manually inserted operational information. Total count data with its higher count rate were normally recorded every 0.5 seconds whereas the three radioelement windows were recorded every 2.5 seconds.

In 1978 the spectrometer system was upgraded to incorporate a NOVA minicomputer for the recording and analysis of 256 channels of gamma-ray information from 0 to 3.0 MeV (Bristow, 1980). The package of twelve cylindrical detectors was also changed to twelve prismatic detectors 10.2 x 10.2 cm (4 x 4 inches) in cross-section which were 40.6 cm (16 inches) long. This detector configuration has approximately the same volume as that used previously but can be packaged more efficiently. The 256 channels of gamma ray data, recorded once a second, were used for accurate energy calibration of the spectrum by monitoring the position of the prominent potassium peak. The windows shown in Table 2, however, were still used to convert the airborne data to ground radioelement concentration.

PRODUCTION OF MAPS AND PROFILES

In order to relate the airborne count rates from the three windows to radioelement concentrations of the ground, four distinct data processing steps are necessary. These are:

(1) The removal of background radiation.
(2) A spectral stripping procedure.
(3) An altitude correction.
(4) The conversion of the corrected count rate data to ground concentration.

In any airborne radioactivity measurement three sources of background radiation exist:

(a) The radioactivity of the aircraft and its equipment.

(b) Cosmic radiation.

(c) Radioactivity in the air arising from daughter products of radon gas in the uranium decay series.

The technique adopted by the GSC has been to remove the effect of these three sources of background radioactivity simultaneously, by utilizing measurements over water. Provided the water body is sufficiently wide and deep the radioactivity measured will be the total background contribution from all three sources. Fortunately, in most of Canada, lakes are abundant and the background values can be updated frequently during the course of the survey.

Due to Compton scattering in the ground, the air, and in the detector, gamma rays originating from one particular radioelement may be detected in any of the three windows. In order to correct for this "cross-talk", a spectral stripping procedure must be carried out. This is achieved through the use of large radioactive concrete calibration sources which were constructed at Uplands Airport in Ottawa (Grasty and Darnley, 1971) and from which the gamma ray spectrum of the respective radioelement may be determined. The stripping procedure used to derive the corrected counts in each window that originate from the respective radioelement has been described by Grasty (1976).

The number of gamma rays detected per second in each window is dependent on the altitude of the aircraft above the ground. In the range of altitudes normally encountered in airborne survey operation the stripped and background corrected count rates in each window, N, is found experimentally to be related to the aircraft altitude, h, by a simple exponential expression of the form:

$$N = Ae^{-\mu h} \qquad (5)$$

where A and are constants (Darnley et al, 1968; Kogan, 1971; and Burson, 1973).

Figure 3 shows the stripped and background corrected potassium count rate variation with aircraft altitude over the GSC airborne gamma-ray spectrometer calibration range at Breckenridge about 30 km west of Ottawa (Charbonneau and Darnley, 1970). The exponential curve given by equation (5), is also shown. This curve is used to correct the count rates in each window for deviations from the planned survey altitude.

From the measured ground radioelement concentration of this calibration range the sensitivity of the airborne gamma-ray spectrometer in terms of counts per unit time per unit concentration of potassium, uranium, and thorium may be determined at the nominal survey altitude allowing the ground concentration to be evaluated over an unknown area.

In order to determine the ground level concentrations of the GSC airborne calibration range, 70 soil samples were collected at seven sites along the 10 km length of the strip, in the summer of 1973, (Grasty, 1975). These samples were then sealed in metal cans and stored for 4 weeks to allow the gamma-ray activity of bismuth-214 to reach equilibrium and analysed in the laboratory by gamma-ray spectrometry for potassium, uranium and thorium. These results are presented in Table 2, together with the results of a detailed ground gamma-ray spectrometer survey carried out with a portable spectrometer calibrated on the radioactive concrete calibration pads at Uplands Airport in Ottawa (Charbonneau and Darnley, 1970).

In the case of potassium and thorium the field and laboratory measurements show good agreement. However, the uranium laboratory assays are considerably higher than the field measurements. These results can readily be explained by a loss of radon-222 from the surface soil of the calibration strip and a corresponding decrease in the bismuth-214 activity. Radon losses of up to 40 percent are not unusual for the clay material characteristic of the Breckenridge strip (Barretto et al, 1975)). In the case of the laboratory assays, the radon and associated bismuth-214 activity is allowed to build up and reach equilibrium because the cans are sealed and the radon gas cannot escape.

Additional confirmation of significant radon loss from the soil surface is provided in Figures 4 and 5. These figures show the background corrected and stripped potassium, uranium and thorium count rate variation over the strip for 8 flights carried out at different times over the past 3 years. The average thorium and potassium count rates show a predictable linear relationship due to changes in the moisture content of the soil. A ten percent increase in soil moisture content will decrease the gamma-ray flux by approximately the same percentage (Kogan et al, 1971). The stripped potassium and thorium count rates had their lowest values when the soil was completely saturated and pools of water were observed along the strip. In contrast with the behaviour of thorium and potassium, the highest stripped uranium count rates were observed when the soil was saturated with water (Figure 5). These results can be readily explained because water in the soil reduces the emanation of radon from the ground allowing the bismuth-214 gamma-ray activity to increase. Similar results have been observed by Stromswold (1978) when the concrete calibration pads at Grand Junction, Colorado, were saturated with water.

In calculating the exposure rate above the ground due to gamma-rays emitted by the uranium series it is necessary to know the gamma-ray activity of the ground in the state of equilibrium existing at the time of the measurement. Due to variations in the emanation of radon, this is best provided by field gamma-ray spectometer measurements and not by laboratory assays on sealed samples which can indicate an artificially high gamma-ray activity. The equivalent uranium concentration of the test strip was therefore assigned the field value of 0.5 +/- 0.1 ppm (Table 2). The thorium and potassium laboratory analyses were considered more reliable than the field measurements because of the greater number of samples analysed (Table 2). In addition, both thorium and potassium measurements are unaffected by sealing the soil samples in the laboratory sample containers.

Early in the airborne gamma-ray program, ground and airborne data were compared over many different rock types in the Bancroft area of Ontario that covered a large range of radioactive concentrations (Charbonneau and Darnley, 1970). A total of 849 ground measurements were carried out with a calibrated portable gamma-ray spectrometer over each rock unit on a regular grid. The airborne measurements were carried out utilizing 3, 12.7x12.7 cm (5x5 inch) sodium iodide detectors mounted in a helicopter which hovered at an altitude of 76 m (250 feet) over each rock unit. The helicopter system was subsequently calibrated by flying over the Breckenridge calibration range, utilizing the assigned concentrations (Table 2).

Figure 6, 7, and 8 show the comparison between the ground and airborne results over the fifteen rock units. The ground and airborne potassium and thorium results show particularly good agreement, whereas the uranium results show somewhat more scatter. However, as is shown later in this paper the gamma-rays originating from uranium generally have only a small contribution to the total gamma-ray exposure rate. Relatively large errors in the airborne uranium measurements therefore have little effect on the calculated total gamma-ray exposure rate.

Based on the potassium, uranium, and thorium count rates recorded over the test strip, the sensitivities of the two airborne systems were evaluated. These results are presented in Table 3, both for the older system utilizing twelve 22.96 x 10.2 cm. (9 x 4 inch) cylindrical detectors, and the new system with the prismatic detectors. The results for the older system represent an average of fifteen separate flights. In the case of the current system, only three flights were used. The flights in early spring and late fall were rejected as the soil conditions at the time were extremely wet and were not representative of the conditions occurring when the ground spectrometer survey was carried out.

THE RELATIONSHIP BETWEEN RADIOELEMENT CONCENTRATION AND EXPOSURE

In order to evaluate the radiation exposure at ground level with a known concentration of the radioelements it is necessary to calculate the energy distribution of the gamma ray flux of each of the three radioelements. This is an extremely complex problem since several hundred gamma ray energies are involved, each with different attenuation coefficients and with multiple scattering occurring both in the ground and in the air. However, with the advent of high speed computers the energy and angular distribution of both the direct and scattered gamma ray component can now be determined. This has been carried out by Beck and his co-workers at the Environmental Monitoring Laboratories in New York (Beck and de Planque, 1968) for the purpose of evaluating the exposure rate from natural gamma radiation and fall-out from nuclear weapons tests. Independently Kirkegaard (1972) has carried out similar calculations to aid in the interpretation of gamma ray surveys for exploration and arrived at similar solutions. Both calculations solve the Boltzmann transport equation for two semi-infinite homogeneous media, one being the ground with a uniform distribution of gamma ray emitters, and the other being the air. Table 4 shows the contribution from potassium, uranium and thorium to the exposure rate 1 m above the ground. The agreement between the results of Beck et al (1972) and Lovborg and Kirkegaard (1974) is a good indication that the energy distribution of the gamma ray flux can be derived reliably.

In the fall of 1981 measurements were taken with a Reuter-Stokes ionization chamber at four sites along the airborne gamma-ray spectrometer calibration range to verify the relationship between radioelement concentration and radiation exposure presented in Table 4. Radiation measurements were also taken from a boat on the Ottawa River nearby, to estimate the combined background radiation exposure due to cosmic radiation, airborne radioactivity, and any small component of instrument background. Soil moisture measurements were taken at each site because of the dependence of the gamma-ray exposure rate on the moisture content of the soil. Utilizing the data, shown in Table 3, and the assigned radioelement concentrations (Table 2) the average exposure rate along the test strip due solely to potassium, uranium and thorium was calculated to be 5.75 +/- 0.12 $\mu R\ h^{-1}$. This calculated value, however, will vary with the moisture content of the soil. Soil moisture measurements taken during the summer of 1978 and 1979 showed that the soil moisture content of the strip had an average value of 27 percent of water by dry weight. The ionization chamber measurements were taken when the soil content, at all four sites, was considerably higher than this average value (Table 5), thereby decreasing the predicted exposure rate. In order to compare the calculated and experimental exposure rate measurements, the measured exposure rates were normalized to the average soil moisture content of the strip of 27 percent utilizing the formula (Kogan, 1971):

$$E_{27} = E_W \times \frac{(100 + 1.11 \times W)}{(100 + 1.11 \times 27)} \qquad (6)$$

where E_{27} and E_W are the exposure rates at 27 and W percent soil moisture (by dry weight), respectively.

The original four exposure rate measurements together with the normalized values are presented in Table 5 after subtracting the average background radiation of 5.0 $\mu R\ h^{-1}$ measured over the Ottawa River.

The good agreement between the predicted and measured exposure rates demonstrates that the concentration of potassium, uranium and thorium in the ground, can be used to provide good estimates of gamma-ray exposure rates at ground level. Good agreement between measured and calculated radiation exposure rates have also been found by Lovborg and Kirkegaard (1974).

THE CONVERSION OF AIRBORNE DATA TO EXPOSURE

Over 200 airborne gamma-ray surveys have been published as GSC Geophysical Series Maps and Open File Reports. Approximately 70 percent of this published data has been produced by survey companies through the Federal-Provincial Uranium Reconnaissance program (Darnley, 1976). Because of some calibration problems with the different aircraft systems involved in the program which is presently under investigation, this paper deals only with airborne data obtained by the two GSC systems previously described. These surveys were generally flown at a nominal survey altitude of 120 meters (400 feet) at a speed of approximately 56 meters per second. Most of the airborne data has been obtained from regional surveys flown at line spacings of 5 kilometers. However, a significant number of surveys have been flown with line spacings as close as 60 meters (200 feet). These more detailed surveys covered known uranium mining areas such as Elliot Lake, Bancroft or Uranium City, or areas of anomalous radioactivity found during the course of the regional surveys. These particular areas, flown because of their anomalous levels of radioactivity are not used in this particular study as they would not be representative of the country as a whole. Table 6 lists the GSC published data obtained from the 5 km. regional survey data analysed in this report. The location of each of the 31 survey areas is shown in Figure 9. The National Topographic Survey (NTS) 1:250,000 sheets covered by these surveys are indicated in Figure 9 and Table 6.

The airborne gamma-ray data can be converted to ground level exposure rates either from the airborne profiles, illustrated in Figure (1), or from the published contour maps of the three radioelements. Because the airborne data were obtained from flight line data 5 km. apart, the ground is more densely sampled along the flight lines than across them. In order to produce contour maps with coherent data from one line to another, the original data were considerably smoothed along the flight lines. With this smoothing the published contour maps, although reflecting regional variations of radioactivity, do not show the same degree of variation as observed in the original profile data. Radioactive highs tend to be suppressed and radioactive lows elevated. In order to preserve the real variations in ground level radioactivity the analysis of the airborne data was carried out solely with the profile data.

Most of the published maps and profiles give the system sensitivities that were used at that time for converting the airborne data to ground concentrations of potassium, uranium and thorium. These sensitivities show some variation from year to year. This variation has now been found to relate to changes in the soil moisture content of the Breckenridge calibration strip at the time the calibration flights were carried out (Figures 4 and 5). The analysis of the airborne data was therefore carried out from the original corrected count-rate data using values of the system sensitivities shown in Table 3.

In establishing statistical parameters, such as the mean and standard deviation of the radiation exposure for each area, it is necessary to consider that a large fraction of Canada is covered by water. We require radiation measurements relating only to the land surface and therefore any overwater values must be removed in the statistical analysis. Studies of airborne profiles have shown that areas of water, such as lakes, bogs and swamps can be successfully identified from the airborne data by monitoring the potassium concentration. Almost all rocks and soils have potassium concentrations well in excess of 0.25 percent potassium (Table 9). Any airborne measurements with potassium concentrations below 0.25 percent were therefore assumed to be associated with overwater measurements and rejected from the analysis.

This technique is used routinely in the processing of the airborne data for automatically identifying areas of water so that the background radiation due to cosmic-rays, airborne radioactivity and the radioactivity of the aircraft and its equipment may be updated.

Histograms of the radiation exposure due to potassium, uranium and thorium and the combined exposure from all three sources were compiled for each survey area shown in Figure 9 and Table 6. These histograms shown in Figures 10 to 16 were then combined to produce histograms for each province (Figures 17 to 19) and for the area of Canada covered by the surveys (Figure 20). They were also used to compute the mean and standard deviations of the radiation exposure for each survey, for each province and also for Canada (Table 7). The mean radiation exposure for each area is illustrated graphically in Figure 21. Various percentiles of the radiation exposure level distributions for each survey area were also computed (Table 8). A percentile, P, is the radiation level below which lie P percentage of the radiation exposure data. For instance these results show that 99 percent of the entire area of Canada surveyed had an exposure of less than 13.30 $\mu R\ h^{-1}$.

A gamma ray spectrometer records gamma rays produced by radioactive decay which is a random process. Consequently some of the apparent variation in the calculated exposure rates for each area are the result of statistical fluctuations due to the counting process. The significance of these statistical fluctuations on the standard deviations shown in Table 7 was calculated from the system sensitivity, calibration constants, counting time, and over water background count rates, assuming each area was uniformly radioactive. It was found that the standard deviation of the exposure rate was over-estimated by more than 10% on only six occasions. These six data sets, indicated in the table, were over-estimated either because they were acquired with a short counting time of one second (two Nova Scotia and one Quebec survey) or because the homogeneous nature of the ground gave a low statistical spread (two Saskatchewan and the Prince Edward Island survey).

THE RELATIONSHIP BETWEEN THE GEOLOGICAL ENVIRONMENT AND EXPOSURE

Gamma-rays have a limited capability to travel through rock or overburden, 90 percent of the signal being derived from the top twenty centimetres (Lovborg et al, 1971). Any airborne gamma ray survey will therefore be influenced mainly by the distribution of rock types at the surface of the earth modified by the effect of overburden or soil which generally covers a large percentage of the bedrock.

Many reviews have been published which give the average levels of potassium, uranium and thorium for different classes of rocks. Table 9, adapted from Killeen (1979), illustrates typical values to be found in the major rock types. An average crustal value is also indicated (Taylor, 1964). Generally all three radioelements vary sympathetically with igneous rocks being more radioactive than metamorphic rocks and sedimentary rocks the least radioactive.

In assessing regional radioactivity patterns it is necessary to consider that the bedrock is generally covered by surface material (overburden). In Canada this overburden is dominated by glacial till which may or may not be capped by soil. This till will have been eroded from outcrops through glaciation and transported some distance before being redeposited on the bedrock surface. Soil has been derived from the till or exposed bedrock by weathering processes. The relationship between the radioelement concentration of the glacial till and the bedrock was a fundamental concern in considering the potential usefulness of regional gamma-ray spectrometer surveys because the airborne signal would originate from the near surface material which would be generally derived from glacial till.

To investigate the relationship between the radioactivity of the glacial till and the underlying bedrock, 24 test sites were selected in three areas of the Canadian Shield (Charbonneau et al, 1976). These sites averaging a few square kilometres in area covered a range of rock types, with associated glacial till cover. In all, more than 2,500 in-situ measurements of potassium, uranium and thorium were made with a calibrated portable gamma-ray spectrometer. Average values were calculated for the radioelement concentrations of the bedrock and the overburden. Exposure rates were then determined using data from Table 4 and the radioelement concentrations calculated from the spectrometer count rates. Figure 22 illustrates the sympathetic relationship that exists between exposure rates for the till and the bedrock. The 24 sites are ordered by increasing potassium concentrations. The rock type code on the bottom is keyed to Table 9, AI for example being acid intrusive rock (granite).

The results in Figure 22 show that the overburden reflects the radioactivity of the underlying bedrock. This is because the overburden is composed of a substantial percentage of locally derived material (Shilts, 1976). Similar observations on the local nature of the overburden have been observed by Pitkin (1968) in the U.S.A. and by Pertunen (1977) in Finland. In areas where the bedrock has high levels of radioactivity, Figure 22 shows that the associated overburden also has high radiation levels. Similarly, low levels of bedrock radioactivity correspond to low levels of radioactivity in the overburden. The overburden has the effect of reducing the amplitude of the variations in the radioactivity of the underlying bedrock.

As can be seen from Table 9 - acid intrusive rocks, (granites) are generally the most radioactive rock types. However, the table shows that sedimentary and metamorphic rocks can also have high radioactivity levels. Figures 23 and 24 show the high exposure rates of most of the granites of Nova Scotia. There is a close correlation between areas above 4 μR h^{-1} and the granites of Devonian - Carboniferous age. Although granites are generally the main rock types underlying any broad radioactive anomaly not all granites have high levels of radioactivity as indicated by the Precambrian granites of Cape Breton Island which do not show an anomalous response.

Figure 25 is a preliminary compilation of airborne data taken from Charbonneau (1982) and shows the distribution of uranium. For this particular map, data produced by survey companies through the Federal - Provincial Uranium Reconnaissance Program (Darnley, 1976) were also included. The sympathetic relationship that generally exists between potassium, uranium and thorium (Fig. 21), would suggest that an exposure rate map would show similar patterns. Many of the more radioactive areas have been investigated on the ground and without exception have been found to be related to granitic rocks with concentrations of uranium and thorium that are several times the crustal average.

Although the underlying rock type is the prime factor that governs the radiation levels at the surface of the ground, the soil type, percentage of rock outcrop, and vegetation cover can also exert a modifying influence over a particular area.

Seldom is pure rock or glacial till exposed at the surface over a wide area without some soil being developed. North of the tree line the soils are poorly developed with little organic content. These soils will be expected to bear a close relationship to the radioactivity of the underlying bedrock. South of the tree line the soils usually have an organic capping and can be water saturated for long periods of time thereby reducing the surface radioactivity. The trees themselves also attenuate the gamma radiation signal from the ground resulting in a somewhat lower estimate of ground radioactivity than would otherwise be observed.

Much of the Northwest Territories and northern Manitoba are above the tree line, have poorly developed soils and a high percentage of rock outcrop. These areas have the highest levels of radioactivity in Canada. While this is principally because the underlying granitic rocks have above average radioactivity, it must also be because of differences in soil type and percentage of rock outcrop as well as the lack of vegetation.

The histograms in Figure 10 through 20 show the distribution of exposure rates from potassium, uranium and thorium and total exposure rate for each survey area. Since most rock types cover a range of concentrations which in most cases overlap, it is frequently impossible to see peaks in the histograms corresponding to the major rock units in a survey area. The effect of overburden also tends to smear differences in the radioactivity levels that may exist between the different rock units. The separation of different rock units can best be seen in map form because of the spatial relationship between the geology and the airborne data which is not considered in the histograms. However, in certain areas where the geology is simple, the histograms can distinguish between rock units which have distinct radioactive signatures. For example, in Figure 19 the histograms of the potassium and total exposure show two distinct peaks for the province of Saskatchewan. The lower peak relates to the Athabasca sandstone which is extremely low in radioactivity, as indicated in Table 7 for sheets 74F and 74G. The upper peak in the histogram relates to crystalline basement rocks which are substantially more radioactive. Similarly in Figure 10, for the Nova Scotia sheets 21A, 21B and 20O, the two peaked histogram results from the difference in radioactivity of granite and metamorphosed sedimentary rock.

FACTORS MODIFYING AIRBORNE EXPOSURE MEASUREMENTS

There are several correction factors that must be applied to the exposure rate estimates from potassium, uranium and thorium as shown in Table 7 to convert them to an average annual outdoor value for the Canadian population.

These factors relate to:-

(a) the population distribution,
(b) the effect of snow,
(c) seasonal soil moisture variations and
(d) the effect of vegetation

(a) The Population Distribution

The data presented in this report was gathered from airborne surveys flown at an average altitude of 120m.

Because of the limitations of flying at these low altitudes over built-up areas, most of the areas flown were relatively unpopulated and represent only a small fraction of the populated areas of Canada. In addition, a large percentage of the data was gathered in the Northwest Territories where the average exposure was significantly higher than the more populated provinces, e.g: Quebec and Ontario (Table 7). Data on the population of each province, taken from the 1981 Canadian Almanac (Walters, 1981), and presented in Table 10, were used to compute a population weighted average from the average exposure rate for each province (Table 7). This reduced the average exposure rate of all the airborne measurements from 4.4 +/- 2.9 $\mu R\ h^{-1}$ to a population weighted average of 3.2 +/- 2.0 $\mu R\ h^{-1}$.

It is also necessary to consider whether the estimates for each province are representative of the areas where the majority of the population resides. For instance, most of the airborne data was gathered over the Canadian Shield whereas most of the population is concentrated in areas suitable for cultivation which are geologically quite different, mainly being flat-lying sedimentary rocks.

A detailed comparison of the Shield and sedimentary areas of southern Ontario was carried out using data published by Loijens and Grasty (1973). These data were used as base levels of radioactivity for an airborne gamma-ray snow survey which covered a large part of the populated areas of south-western Ontario as well as a significant area of the Shield. When allowance was made for the attenuation of the airborne signal by the forest cover of the shield (estimated to be 15 percent) no significant different was found between the radiation levels on or off the Shield.

The similarity of these two quite different geological areas was somewhat unexpected as the Shield was originally believed to be more radioactive. However, further investigation showed that the clays and shales commonly found in southern Ontario have relatively high potassium contents between 3 and 5 percent (Guillet, 1977). In addition, although the Shield has rocks such as granites and pegmatites, some of which have above average radioactivity, these tend to be spatially restricted in distribution. To a large extent the Shield area of southern Ontario is covered by a variety of metamorphosed igneous and metamorphic rocks which are not particularly radioactive. Furthermore, the poor drainage of the Shield also produces large areas of bog and swamp which further reduces its level of radioactivity.

Additional confirmation of the similarity of the two areas may be found from the results presented in Table 7.

The Ontario sheet 31F for Pembroke lies totally within the Canadian Shield and was found to have a similar but slightly lower exposure than the adjacent map sheet 31C for Kingston (Table 7), which is split roughly equally between the Canadian Shield and sedimentary rocks typical of populated areas in Ontario. These results strongly indicate that at least for Ontario the radioactivity levels of Table 7 are representative of the populated areas.

(b) The Effect of Snow

Snow reduces the radiation exposure at the surface of the ground. The attenuation of the radiation depends not on the depth of the snow but on its water content. This attenuation is the basis of an airborne technique to measure the water equivalent of the snow cover and has been widely used in various parts of the world (Kogan, 1971; Grasty, 1973). Because of the difference in the energy spectrum of the gamma-radiation from potassium, uranium and thorium, the attenuation of the ground level exposure rate depends to some extent on the proportions of the three radioactive elements. Lovborg and Kirkegaard (1975) have shown that over material of typical granite composition, 8 cm of water will reduce the exposure rate by approximately fifty percent.

A large part of Canada is snow covered for several months each year. Since the data presented in this report were gathered in the summer months, the average annual outdoor exposure rate will be lower than shown in Table 7. In theory, to calculate the effect of snow, the average snow-water equivalent on the ground must be determined for each province whenever the amount of snow changes. This is because the exposure rate does not vary linearly with snow-water equivalent but follows an approximately exponential curve. However the amount of snow on the ground varies considerably from year to year and in addition it has a relatively minor effect (20 percent) on the exposure rate. It can therefore be assumed that the total amount of snow on the ground during the winter months can be averaged over an entire year and its effect need not be evaluated at different times throughout the winter.

Information on snowfall is readily available, however data relating to the water content of the snow on the ground is limited. The Great Lakes/St. Lawrence Study Office of Environment Canada gathers snow-water equivalent information from a variety of sources each winter to predict the snow-water content of the Great Lakes drainage basin. Since most of the population of Canada resides in Quebec and Ontario (Table 10), data for the Great Lakes basin were used to compute the effect of snow on the average outdoor exposure rate.

This data showed that for the first week of January, 1981, the drainage basin had a 9 year mean water-equivalent content of 41 mm. Together with the values for February and March of 71 and 74 mm respectively, this corresponds to an average annual snow-water equivalent of 15.5 mm. No data was available for December and April. Based on the variation of exposure rate with elevation above the ground, presented graphically by Lovborg and Kirkegaard (1975), it was estimated that this 15.5 mm of water will reduce the exposure rate at ground level by 16 percent. When an additional reduction is made for the presence of snow in December and April the annual outdoor population weighted average exposure rate will be reduced by approximately 20 percent from its measured summer value.

(c) Seasonal Soil Moisture Variations

The results presented in this report show significant variations in radioactivity from area to area, mainly depending on the particular underlying geological formation. However, individual radioactivity measurements can vary significantly with time, mainly through changes in the moisture content of the soil. A 20 percent increase in soil moisture (e.g. 20 to 40 percent) is not uncommon and will in theory decrease the gamma-radiation at the soil surface also by about 20 percent (Equation 2). Variations of this magnitude have been observed in the gamma-ray count rate from both potassium and thorium over the Breckenridge calibration strip (Figure 4) which are undoubtedly related to soil moisture changes.

The effect of soil moisture on the gamma radiation from the uranium series is more complex than it is for potassium or thorium, since a water saturated soil can inhibit the emanation of radon thereby increasing its gamma-ray activity. For our particular calibration strip the gamma-ray activity from uranium can increase by almost 50 percent when the soil becomes saturated (Figure 5), whereas the potassium and thorium activities show a much smaller decrease. The uranium series generally contributes only about ten percent to the total exposure (Table 7) and therefore the effect of large fluctuations in the activity from the uranium series does not have a great effect on the total exposure. In addition, the test strip consists of soil of clay composition, which is known to be a high emanator of radon (Barretto et al, 1975). In other areas with soils of more sandy composition, and lower emanation rates, the variation in gamma-ray activity from the uranium decay series is expected to be less.

Since no data on the moisture content of the soil was obtained during the airborne data collection period, the effect of soil moisture puts a limit on the accuracy of any one particular airborne measurement. The airborne measurements represent many hundreds of days of flying and therefore the average exposure rate for Canada will closely represent the average soil moisture conditions over the summer months when the surveys were carried out. In the summer the soil will on average have a lower soil moisture content than it will for the remainder of the year.

In order to estimate the effect of seasonal soil moisture fluctuations on the calculated gamma-ray exposure rates, we have analysed over 1000 soil moisture measurements reported for 24 different sites in southern Ontario. These measurements were carried out principally during the growing season in the years 1966 to 1968 by the Ontario Agricultural College of the University of Guelph (Selirio et al, 1978).

Table 11 shows the monthly average soil moisture values for the top 15 cm of soil. The top layer of soil is the region of interest since most of the gamma radiation measured at the surface originates here. No data was available for the months of January, February and March. We have assumed that the soil moisture content does not change in January and February from its December value because the ground is frozen. In March, the ground would normally be saturated because of snow-melt and the thawing ground. This saturated value was estimated from the average maximum soil moisture content of the 24 sites.

Considering the number of measurements involved the results clearly show the soil moisture changes that occur throughout the year. Very similar results have been obtained from water balance calculations carried out by the Atmospheric Environment Service of Environment Canada using 30 year mean air temperature data to determine potential evapotranspiration (Thornthwaite and Mather, 1957). Together with precipitation data, variations in the water stored in the soil were then calculated. The tabulated water balance data, measured as millimeters of water, were converted to soil moisture content, by making simple assumptions of generally observed maximum and minimum soil moisture values. In the calculation we assumed that the water stored in the soil was proportional to its soil moisture content.

Table 11 shows that in the four summer months from June to September, when most of the surveys were carried out, the average soil moisture content is 31.5 percent. This compares to an annual average of 37.8 percent. Using equation (2) this change in soil moisture will correspond to a change in exposure rate of approximately 5 percent. Consequently the summer exposure rate data, shown in Tables

7 and 8 must be decreased by 5 percent when average annual values are considered.

(d) The Effect of Vegetation

In utilizing the results presented in this paper it is important to consider the effect of trees on the airborne measurements of gamma radiation, since they absorb gamma radiation from the ground. An additional complicating factor arises because the aircraft radar-altimeter is reflected from the forest canopy and not from the ground below the trees. The aircraft altimeter will therefore register an apparent altitude above the ground which is less than the true value. The error in the altitude will depend on the height of the trees and the density of the forest cover. Over a forested area, the effects of gamma ray attenuation and the radar-altimeter error both result in an estimate of ground radioactivity which is lower than the true value.

The quantity of bio-mass in a forest can vary widely. Rubin et al (1979), in a study of the effects of vegetation on the uranium spectrum reports that 50% of forests in the U.S. have an above ground bio-mass between 0 and 3.5 gm/cm^2. Gordon (1981) gives values between 1.0 and 1.6 gm/cm^2 for three average stands of trees in northern

Ontario. Using a bio-mass value of 1.3 gm/cm^2 and an air density of 0.001293 gm/cm^3 this corresponds approximately to a 10 m equivalent layer of air. The error associated with the altimeter was estimated to be 10 m. The aircraft is therefore flying at an equivalent elevation above ground which is around 20 m higher than registered by the radar altimeter.

Based on the attenuation of gamma-radiation with aircraft altitude (Figure 3 and Equation 5), and using attenuation coefficients given by Glynn and Grasty (1980) 20 m of air will reduce the calculated potassium and thorium count rates and their associated concentrations by 16 and 13 percent respectively. Since potassium and thorium have an approximately equal contribution to the total exposure, the average outdoor exposure rates presented in Table 7 and 8 must be increased by approximately 15 percent.

ANNUAL OUTDOOR DOSE FROM TERRESTRIAL RADIATION

(a) Calculation

In computing an average annual outdoor dose-equivalent from terrestrial radiation, it is necessary to consider the effects described in the previous section, the magnitude of which are given in Table 12. This table shows that over an entire year the exposure rate of the average Canadian must be reduced by 10 percent from the population weighted average of 3.2 +/- 2.0 $\mu R\ h^{-1}$ (Table 10) as calculated from the airborne data collected in the summer months. With this figure of 10 percent the average annual population weighted outdoor exposure rate from terrestrial radiation reduces to 2.9 +/- 1.8 $\mu R\ h^{-1}$. This average annual outdoor exposure rate corresponds to an annual exposure of 25 +/- 16 mR.

Equation 4 derived earlier in the section on Dose Relationships can be used directly to convert this annual exposure to annual whole-body dose-equivalent using a geometry factor, g, of 0.75. This factor accounts for the self-shielding effect of the body. The average annual outdoor dose-equivalent from terrestrial radiation was calculated as 18 +/- 11 mrem (180 +/- 110 µSv).

(b) Discussion of Results

UNSCEAR (1977) have estimated the world average outdoor absorbed dose rate in air to be 4.5 rad h^{-1} (0.045 Gy h^{-1}). From equation (2) this corresponds to an exposure rate of 5.2 $\mu R\ h^{-1}$. The UNSCEAR value was based on large area surveys from ten different countries for which the exposure rate ranged from 4.1 to 10.5 $\mu R\ h^{-1}$ (see Table 13). Our outdoor summer value for Canada of 2.90 $\mu R\ h^{-1}$ is considerably less than the estimated world average and is also significantly lower than any of the ten countries.

To put the Canadian results into perspective, it should be noted that the majority of the population of Canada live in Ontario and Quebec which are significantly lower in radioactivity than Manitoba and the Northwest Territories (Table 7). If the Canadian population was concentrated in either of these two areas with higher radioactivities our results would be quite different and much closer to the estimated world average. They would, however, still be among the lowest.

The fact that Canada has a low average level of radioactivity can be explained in terms of a simple model of crustal evolution.

There is substantial evidence of a general decrease of continental heat flow with the age of crustal material, the older Precambrian Shield showing very low heat flows (Vitorello and Pollack, 1980; Hanza and Verman, 1969, etc.). This heat is generated by the decay of radioactive material from within the crust. Additional sources of heat originate from deep beneath the continent, in its roots, and perhaps from the earth's core. From a knowledge of the surface radioactivity, the observed heat flow is found to be too low to be explained by a uniformly radioactive crust. It can only be explained by a decrease of radioactivity with depth below the crustal surface (Lachenbruch, 1970). In time this surface is eroded, removing the more radioactive material, and exposing material with a lower radioactivity. Based on a simple model for this decrease of radioactivity with depth, Vitorello and Pollack (1980) have estimated that the heat flow originating from within the continental crust can vary by as much as a factor of two, depending on the age of the rocks. The radioactivity of the surface rocks would be expected to show a similar variation.

It should be emphasized that the relation of the age of the rock to its radioactivity is only a general one. Rao and Jessop (1975) have shown that the radiation levels and associated heat flow is a function not only of the age of the rock but also of the particular rock type involved (Table 9). However, even without a detailed investigation of the particular rock types covered by the surveys listed in Table 16, the majority of the results are not unexpected.

The European countries shown are underlain by rocks which are much younger than those in the areas surveyed in Canada. These countries show the highest exposure levels.

India and Canada are the only countries surveyed which have large areas covered by very much older Shield material. Both India and Canada show the lowest exposure rates. With this model of the decrease in radioactivity with depth in the crust, these results would be expected. It is also interesting to note that the older Precambrian granites of Nova Scotia are low in radioactivity compared to the younger Devonian Carboniferous granites (Figures 23 and 24).

ADDITIONAL COMPONENTS OF OUTDOOR RADIATION

In estimating the average annual outdoor dose equivalent from all sources of natural radiation, three additional sources must be considered. These are:

(a) cosmic radiation,
(b) airborne radioactivity,
(c) the internal radioactivity of the body,

(a) Cosmic Radiation

A large component of the radiation dose to the human population arises from high energy cosmic radiation entering the earth's atmosphere. The primary cosmic rays, mainly consisting of high energy protons, interact with atomic nuclei to produce electromagnetic radiation and secondary particles such as pions, muons, neutrons and electrons. Below an altitude of 5 km most of the radiation dose arises from muon collision electrons and muon decay electrons (O'Brien, 1975). A small non-ionizing component of the radiation dose arises from neutrons.

The cosmic ray intensity shows small fluctuations of around +/- 5 percent related to the phase of the 11 year solar cycle. It also varies to some extent with geomagnetic latitude because of the screening effect of the earth's magnetic field which is greatest at the lower latitudes. Since the atmosphere attenuates the cosmic ray flux, the cosmic ray intensity increases with altitude, doubling approximately every 2000 m. Changes in barometric pressure and the associated differences in atmospheric attenuation also cause small fluctuations of a short-term nature. Solar flares can result in increases in cosmic ray activity. However solar cosmic rays have relatively low energy and rarely cause any significant increase in the radiation dose at the earth's surface (I.C.R.P., 1966).

Estimates of the cosmic radiation levels in Canada were derived from theoretical data published by O'Brien (1975) and O'Brien and McLaughlin (1972). These data, presented conveniently in the form of tables show good agreement with experimental measurements of Neher (1967) and Lowder and Beck (1966). More recent measurements at the Environmental Measurement Laboratory in New York also show reasonable agreement with the theoretical data (Volchok et al, 1981). The rather high experimental values of George (1970) which are frequently incorporated into cosmic ray dose estimations (Oakley, 1972) are believed to be due to contamination from atomic weapons fallout (Liboff, 1975).

Population data from the 1981 Canadian census for 24 metropolitan areas representing 56 percent of the population were used to derive the average longitude, latitude and elevation above sea level of the Canadian population. This was found to be at an elevation of 170 m, and located at 46°N and 88°W in Wisconsin, U.S.A.! From the position of the geomagnetic pole at 78°N and 69°W this position corresponds to a geomagnetic latitude of 57°N which is close to the 55°N for which theoretical data was calculated by O'Brien (1975).

At Canadian latitudes, Carmichael and Bercovitch (1969) have found that both the neutron and muon fluxes at sea-level vary by less than 1 percent. The world-wide surveys of Millikan and Neher in the early 1930's also found that north of latitude 35 degrees the cosmic ray fluxes are constant over the entire North American continent (Millikan and Neher, 1936). Recent measurements reported by the Advisory Committee for Radiation Biology Aspects of Supersonic Transport (ACRBAST, 1975) also show very little variation of radiation dose for geomagnetic latitudes between 37° and 58°N at an altitude of 3 km. Consequently O'Brien's data at 55° can be used reliably to evaluate the population dose at different elevations in Canada. These data, representing the ionizing component of cosmic radiation, are presented in Figure 26 and is the mean of the tabulated values at solar minimum and solar maximum.

At sea-level the dose-equivalent rate is approximaely 29 mrem y^{-1} (290 μSv y^{-1}) but reaches a value as high as 43 mrem y^{-1} (430 μSv y^{-1}) for Banff, Alberta at an altitude of 1400 m. The result at sea-level of 29 mrem y^{-1} (290 μSv y^{-1}) compares favourably with the values of 28.6 and 27.6 reported by Shamos and Liboff (1972) and Lowder and Beck (1966) respectively. The population weighted average cosmic ray ionization dose-equivalent rate for the mean population elevation of 170 m is 30 mrem y^{-1} (300 μSv y^{-1}), only slightly higher than the sea-level value. It should be pointed out that these are outdoor values and do not include any shielding effect from buildings.

The data presented in Figure 26 can also be used to estimate short term fluctuations due to barometric pressure changes. These pressure changes would generally not be more than +/- 2 percent corresponding to an equivalent elevation change of around 150 m. Such an elevation change would vary the dose-equivalent by only +/- 1 mrem y^{-1} (10 µSv y^{-1}).

The neutron absorbed dose rate in air is negligible compared to that from charged particles. However, in tissue which has a high proportion of hydrogen atoms, the neutron absorbed dose rate must be taken into account. For consistency, we have also adopted the calculated values of O'Brien and McLaughlin (1972) for the neutron dose rate at sea-level. Their value of 2.2 mrem y^{-1} (22 µSv y^{-1}) is in good agreement with the experimental value of Hajnal et al (1971) and also agrees closely with the figure adopted by UNSCEAR (1977).

The total outdoor dose-equivalent rate for all components of cosmic radiation is therefore estimated to be 32 mrem y^{-1} (320 µSv y^{-1}). This figure will vary by approximately 5 percent (1.5 mrem y^{-1} or 15 µSv y^{-1}) depending on the solar cycle. Similar short-term fluctuations of around 1 mrem y^{-1} (10 µSv y^{-1}), due to barometric pressure changes will also occur.

(b) Airborne Radioactivity

Atmospheric radon daughter products also have a small contribution to the radiation dose at ground level. This contribution shows both seasonal and diurnal variations (Gold et al, 1964).

During the course of the routine airborne surveys, the GSC has carried out many measurements of atmospheric background over lakes and large rivers. From these background measurements we have calculated that under unusual meteorological conditions, such as temperature inversions, the exposure rate from radon daughters can be as high as 2.0 μR h^{-1}. Analysis of data from Nova Scotia showed a mean exposure rate of 0.08 μR h^{-1} with values reaching 0.34 μR h^{-1}. In these calculations the aircraft was assumed to be flying in an effectively infinite homogeneous source of radiation and therefore receiving the same radiation from above and below the aircraft. In this situation the exposure rate at ground level and also the counts that would be observed in the bismuth window at ground level will be half their value at the aircraft altitude. The bismuth-214 window count rates can then be converted to an equivalent uranium concentration and associated exposure rate using the relationships indicated in Tables 3 and 4. We also calculated a similar average annual value of 0.07 $\mu R\ h^{-1}$ from radon concentration data presented by (Gold et al, 1964) using the relationship between total body exposure and radon concentration given by Kocher (1980).

In comparison to terrestrial and cosmic radiation the contribution of airborne radon to the whole-body dose is insignificant and can be neglected. However, as we have previously pointed out we are not considering the effects of alpha and beta particles which are significant in terms of radiation dose to the lung and associated organs.

(c) The Internal Radioactivity of the Body

This paper is principally concerned with external sources of radiation. However, a significant fraction of the radiation dose to the human body arises from naturally occurring radioactive elements principally potassium-40, carbon-14 and to a small extent rubidium-87 which are taken into the body. For the sake of completeness we include some basic data on these sources of internal radiation.

The dose to specific organs of the body can vary considerably. In this report only the average radiation dose received by the whole body is considered. These data have been taken directly from the UNSCEAR report (UNSCEAR,

1977) and for more complete information on the dose received by various organs, the reader should refer to this report.

Potassium-40 is the major naturally occurring source of internal radiation dose. The dose rate from potassium-40 can be calculated from its isotopic abundance and the concentration of potassium in human tissues. Similarly the dose from rubidium-87 may be calculated from their concentration in various organs of the body and this has been carried out for the International Commission on Radiological Protection Reference Man (ICRP, 1975).

When primary cosmic-ray particles enter the earth's atmosphere they may undergo reactions with atomic nuclei present in the air producing neutrons, protons, pions and kaons. Many of the secondary particles have sufficient energy to initiate further nuclear reactions with nitrogen and oxygen nuclei. Very little of the external dose from the natural background is contributed by cosmogenic nuclei produced through these reactions. However, some of the internal radiation dose arises from carbon-14 which is produced from the capture of neutrons by nitrogen-14 and eventually taken into the body.

The internal dose from all three sources of radiation are presented in Table 14 which shows that potassium-40 is by far the greatest source of internal radiation dose.

The average annual Canadian outdoor dose-equivalent from all sources of natural radiation is summarized in Table 15.

ESTIMATION OF ANNUAL DOSE-EQUIVALENT

In estimating the annual dose-equivalent, it is necessary to consider that most people spend a large percentage of their time indoors where the building material acts as both a source of radioactivity and a shield. Estimates of indoor dose can in theory be derived from the radioactivity of the various building material and its configuration. Unfortunately, very little of this information is available. We have therefore followed the procedure adopted by UNSCEAR (1977) which is to estimate the average indoor dose from the outdoor value using a conversion factor which at least makes some allowance for the effect of buildings.

From an analysis of published data UNSCEAR (1977) have concluded that the world average indoor exposure rate from terrestrial radiation is 18 percent higher than it is outside. This reflects the fact that the average person spends a large percentage of his time in apartment blocks or office buildings. For these buildings the radiation levels would be significantly higher than for buildings of wood frame construction where the building materials (wood and plaster) are generally low in radioactivity compared to brick and concrete. An apartment building or office block would also be expected to have higher radiation levels than a wood frame building, purely from geometrical considerations. This is because in such a building the lower floors are receiving radiation from building material in all directions rather than just from below.

For lack of any other information and from our own limited experience we have accepted the UNSCEAR figure of 18 percent as being the increase of the indoor radiation levels over the outdoor terrestrial values.

The population weighted summer outdoor exposure rate of 3.2 +/- 2.0 $\mu R\ h^{-1}$ (Table 10) does not include the attenuation of the airborne signal by vegetation. For the Provinces of Ontario and Quebec, where the majority of the population resides, this amounts to 15 percent (Table 12), resulting in an average outdoor summer exposure rate of 3.7 +/- 2.3 $\mu R\ h^{-1}$. Using the indoor to outdoor conversion factor of 1.18 we arrive at a figure of 4.4 +/- 2.7 $\mu R\ h^{-1}$ for the average indoor exposure rate. No correction has been applied for snow or soil moisture variation throughout the year since inside a building these effects will be minimal. Using equation 4 the indoor exposure rate of 4.4 +/- 2.7 μR h^{-1} corresponds to an annual indoor dose-equivalent of 27 +/- 17 mrem (270 +/- 170 μSv).

Oakley (1972) has estimated that the U.S. population spends 95 percent of their time indoors. It would seem reasonable that the living habits of the average Canadian is not substantially different from their neighbours nearby. The annual whole-body dose is therefore essentially controlled by the indoor dose and can be assumed to have the same value of 27 +/- 17 mrem (270 +/- 170 μSv).

From the relationship between cosmic-ray dose-equivalent and elevation above sea-level (Figure 15) it can easily be calculated that between 200 and 300 gm/cm^2 of structural shielding will reduce the outdoor cosmic-ray intensity by 50 percent. This amount of shielding is not unreasonable for a large apartment block or office building. NCRP (1975) have taken a figure of 10 percent as an average attenuation factor for outdoor cosmic radiation values. For lack of any other information we have also adopted this figure of 10 percent which reduces the annual cosmic-ray dose-equivalent to 29 +/- 3 mrem (290 +/- 30 μSv). Table 16 shows our estimated average annual whole-body dose-equivalent from all sources of natural radiation to be 75 +/- 17 mrem (750 +/- 170 µSv).

Although the effect of buildings must remain one of the largest source of error in estimating the annual radiation dose, it should be noted that terrestrial radiation contributes less than 30 percent of the annual dose (Table 15). Consequently an extreme error of 30 percent in the conversion factor between indoor and outdoor exposure will result in an error of less than ten percent in the estimated annual dose.

SUMMARY AND CONCLUSION

1) This paper has demonstrated that ground level exposure rates can be measured by airborne gamma-ray spectrometry from the surface concentrations of potassium, uranium, and thorium.

2) Airborne measurements of ground level concentrations were confirmed using calibrated portable gamma-ray spectrometers over 24 test sites.

3) Ground level exposure rates calculated from airborne data recorded over an airborne calibration range were

confirmed by measurements with a Reuter-Stokes Ionization chamber.

4) Airborne surveys flown over 33 areas, representing approximately 900,000 individual measurements, were used to compile information on average ground level exposures in Canada.

5) Large areas of anomalously high radioactivity in Northern Manitoba and the Northwest Territories were generally found to be related to granitic rocks.

6) The increased radioactivity of northern Canada results to some extent from its lack of vegetation, and high percentage of rock outcrop.

7) The lowest radioactive areas were found to be associated with the Athabasca sandstone.

8) Outdoor summer exposure rates from potassium, uranium and thorium for the 33 study areas, had an average of 4.4 +/- 2.9 $\mu R\ h^{-1}$.

9) Of this 4.4 $\mu R\ h^{-1}$, 48 percent originated from potassium, 43 percent from the thorium series, and the remaining 9 percent from the uranium series.

10) The population weighted average summer outdoor exposure rate from terrestrial radiation, was 3.7 +/- 2.3 μR h^{-1}. This is considerably less than the world average of 5.2 μR h^{-1} estimated by UNSCEAR (1977) but can be explained by a simple model of crustal evolution in which the radioactivity of the crust decreases with depth.

11) When the effects of seasonal variations of soil moisture, and the attenuation of the ground radiation by snow are considered, the population weighted average annual outdoor exposure rate from terrestrial radiation is found to be 2.9 +/- 1.8 μR h^{-1}.

12) Including a body shielding factor of 0.75, the average annual Canadian outdoor whole-body dose-equivalent from terrestrial radiation is found to be 18 +/- 11 mrem (180 +/- 110 μSv).

13) Additional components of outdoor annual dose-equivalent arise from cosmic radiation (32 +/- 3 mrem - 320 μSv) and the internal radioactivity of the body (19 mrem - 190 μSv) which give a total outdoor annual value of 69 +/- 11 mrem (690 +/- 110 μSv).

14) Based on a comparison of indoor and outdoor values from world-wide published data, the average annual

Canadian whole-body dose-equivalent from all sources of natural radiation is estimated to be 75 +/- 17 mrem (750 +/- 170 μSv).

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TABLES

1) Spectral Windows.

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3) Airborne gamma-ray sensitivities.

4) Calculated contribution of potassium, uranium and thorium to the exposure rate 1m above the ground.

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13 Average outdoor exposure rates for some countries (UNSCEAR, 1977).

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15) Average annual Canadian whole-body outdoor dose equivalents from natural sources of radiation.

16) Estimated average annual Canadian whole-body dose-equivalents from natural sources of radiation.

TABLE 1

SPECTRAL WINDOWS

Element Analyzed	Isotope Used	Gamma-ray Energy (MeV)	Energy Window (MeV)
Potassium	K-40	1.46	1.37 - 1.57
Uranium	Bi-214	1.76	1.66 - 1.86
Thorium	Tl-208	2.62	2.41 - 2.81
TOTAL COUNT			0.41 - 2.81

TABLE 2

RADIO-ELEMENT CONCENTRATION OF THE BRECKENRIDGE AIRBORNE CALIBRATION RANGE

Type of Measurement	Number of Analyses	Potassium (Percent)		Uranium* (ppm)		Thorium* (ppm)	
Laboratory (Sealed can assay)	70	2.03	0.04	0.92	0.09	7.70	0.28
Field	27	1.8	0.1	0.5	0.1	8.0	0.4
Assigned Value		2.03	0.04	0.5	0.1	7.70	0.26

* Assuming radioactive equilibrium.

TABLE 3

AIRBORNE GAMMA-RAY SPECROMETER SENSITIVITIES

SENSITIVITY (COUNTS/SEC.)

System	Potassium Per %K		Uranium Per ppm		Thorium Per ppm	
12 9 X 4 inch Cylindrical Detectors	78.9	4.2	19.1	4.3	6.1	0.4
12 4 X 4 X 16 inch Prismatic Detectors	90.9	1.8	15.2	0.9	7.0	0.1

TABLE 4

CALCULATED CONTRIBUTION OF POTASSIUM, URANIUM AND THORIUM TO THE EXPOSURE RATE 1 M ABOVE THE GROUND

	Exposure Rate (uR/hr)		
	Lovborg & Kirkegaard 1974	Beck et al 1972	Assigned Value
1% K	1.52	1.49	1.505
1 ppm U*	0.63	0.62	0.625
1 ppm Th	0.31	0.31	0.31

*Assuming radioactive equilibrium.

TABLE 5

GAMMA-RAY EXPOSURE RATES OF THE BRECKENRIDGE CALIBRATION RANGE

		Exposure Rate (μR/hr)	
Site	Soil Moisture (Percent Dry Weight)	Measured	Normalized to a soil moisture content of 27 percent
1	37	5.2	5.6
2	39	5.05	5.6
3	75	4.4	6.2
4	38	5.2	5.7
Average	47	5.0	5.8 0.3

Theoretical Value 5.75 0.12

TABLE 6

5km. AIRBORNE SURVEYS USED IN THE DATA COMPILATION

OPEN FILE NUMBER	DATE PUBLISHED	AREA	NTS SHEETS
270	June, 1975	Burin Peninsula, Nfld.	1N
816	January, 1982	Sydney, N.S.	11K,11F(Part)
429	March, 1977	Annapolis-Shelburne,N.S.	21A,21B,20-O
35411G* 35511G* 35611G* 35821G*	January, 1981	Halifax, N.S.	11D,E,(11F, 21H)Parts
269	June, 1975	P.E.I	11L,21I(Part)
271	June, 1975	Havre St.Pierre, P.Q.	12L
36031G*	August, 1979	Mont Laurier, P.Q.	31J
331	June, 1976	Pembroke, Ont.	31F
428	March, 1977	Kingston, Ont.	31C
262	May, 1975	Blind River, Ont.	41J
329	June, 1976	Ignace,Sioux Lookout, Ont.	52G,J
315	March, 1976	Brochet, Man.	64F
316	March, 1976	Tadoule Lake, Man.	64J
317	March, 1976	Whiskey Jack Lake, Man.	64K
318	March, 1976	Kasmere Lake, Man.	64N
319	March, 1976	Munroe Lake, Man.	64O
309	April, 1976	Reindeer Lake, Sask.	64E
310	April, 1976	Wollaston Lake, Sask.	64L
35672G*	December, 1977	Cypress Hills, Sask.	(72F,G,J)Parts
311	April, 1976	Foster Lake, Sask.	74A
314	April, 1976	Lloyd Lake, Sask.	74F
312	April, 1976	Cree Lake, Sask.	74G
313	April, 1976	Geikie River, Sask.	74H
257	April, 1975	Fond Du Lac East, Sask.	74P,64M
257	April, 1975	Fond Du Lac West, Sask.	74N,O
35574G* 36274G* 36374G*	June, 1979	North Eastern Alberta	74E(Part),L,M
101	July, 1972	Forth Smith, N.W.T.	75D,E,(75L,74M)Par
124	January, 1973	Yellowknife East, N.W.T.	(75L,85I)Parts
124	January, 1973	Yellowknife West, N.W.T.	(85J,85I)Parts
188	April, 1974	Marian River East, N.W.T.	75M,85P
188	April, 1974	Marian River West, N.W.T.	85N(Part),85O
140	April, 1973	Bear Slave East, N.W.T.	86A,H,(B,G)Parts
140	April, 1973	Bear Slave West, N.W.T.	86C,F,(B,G)Parts

* Published as Geophysical Map Series.

Table [illegible] Mean and Standard Deviation of the Summer Outdoor Exposure Rate (Uncorrected for Vegetation)

Area	Number of Measurements	Potassium ($\mu R\ h^{-1}$)	Uranium ($\mu R\ h^{-1}$)	Thorium ($\mu R\ h^{-1}$)	Total ($\mu R\ h^{-1}$)
Nova Scotia - 11K,11F(Part)	33670	1.40 +/- .80	.27 +/- .49*	1.26 +/- .73	2.93 +/- 1.74
Nova Scotia - 21A,21B,200	24673	1.95 +/- .94	.37 +/- .39	1.32 +/- .63	3.63 +/- 1.88
Nova Scotia - 11D,E,(11F,21H)Parts	86090	1.63 +/- .72	.41 +/- .43*	1.41 +/- .60	3.45 +/- 1.52
Quebec - 12L	17307	1.11 +/- .63	.17 +/- .44	.69 +/- .61	1.97 +/- 1.64
Quebec - 31J	25862	1.87 +/- .67	.27 +/- .34*	1.03 +/- .59	3.17 +/- 1.51
Ontario - 31F	25366	1.85 +/- .67	.26 +/- .35	1.08 +/- .58	3.19 +/- 1.55
Ontario - 31C	25048	2.08 +/- .91	.32 +/- .36	1.14 +/- .63	3.54 +/- 1.81
Ontario - 41J	20153	2.02 +/- .83	.42 +/- .43	1.89 +/- 1.36	4.32 +/- 2.52
Ontario - 52G,J	41102	1.42 +/- .72	.21 +/- .34	1.09 +/- .81	2.72 +/- 1.81
Manitoba - 64F	18031	2.31 +/- 1.22	.26 +/- .34	1.74 +/- .95	4.31 +/- 2.39
Manitoba - 64J	19691	2.57 +/- 1.39	.40 +/- .40	2.32 +/- 1.32	5.29 +/- 3.02
Manitoba - 64K	20878	2.93 +/- 1.50	.48 +/- .44	3.06 +/- 1.67	6.47 +/- 3.66
Manitoba - 64N	19852	3.09 +/- 1.46	.60 +/- .49	3.37 +/- 1.69	7.06 +/- 4.09
Manitoba - 64O	18407	2.76 +/- 1.37	.42 +/- .43	2.41 +/- 1.29	5.59 +/- 3.28
Saskatchewan - 64E	16133	2.37 +/- 1.12	.34 +/- .38	1.88 +/- 1.06	4.59 +/- 2.42
Saskatchewan - 64L	18192	2.35 +/- 1.18	.34 +/- .39	2.06 +/- 1.11	4.75 +/- 2.54
Saskatchewan - (72F,G,J)Parts	21556	2.89 +/- .45	.57 +/- .35	2.38 +/- .47*	5.84 +/- 1.23
Saskatchewan - 74A	21478	2.44 +/- 1.00	.35 +/- .36	1.84 +/- .89	4.63 +/- 2.09
Saskatchewan - 74F	15549	.62 +/- .39	.09 +/- .33	.82 +/- .43*	1.53 +/- 1.22
Saskatchewan - 74G	12599	.69 +/- .44	.10 +/- .33	.90 +/- .50	1.68 +/- 1.30
Saskatchewan - 74H	21350	1.56 +/- 1.04	.22 +/- .35	1.61 +/- .91	3.39 +/- 2.13
Saskatchewan - 74P,64M	36573	2.14 +/- 1.03	.31 +/- .40	1.51 +/- .95	3.96 +/- 2.39
Saskatchewan - 74N,O	31420	2.09 +/- 1.13	.34 +/- .45	1.57 +/- 1.05	4.00 +/- 2.47
NWT - 75D,E,(75L,74M)Parts	51431	2.83 +/- 1.33	.51 +/- .50	2.94 +/- 1.44	6.28 +/- 3.99
NWT - (75L,85I)Parts	30789	2.58 +/- 1.26	.70 +/- .57	2.06 +/- 1.33	5.35 +/- 3.32
NWT - (85J,85I)Parts	32573	2.52 +/- 1.12	.72 +/- .53	2.06 +/- 1.26	5.30 +/- 2.83
NWT - 75M,85P	28759	2.55 +/- 1.24	.62 +/- .59	2.41 +/- 1.61	5.58 +/- 3.46
NWT - 85N(Part),85O	31422	2.74 +/- 1.24	.61 +/- .50	2.79 +/- 1.47	6.13 +/- 3.11
NWT - 86A,H,(B,G)Parts	30619	2.27 +/- .81	.37 +/- .36	2.44 +/- 1.21	5.08 +/- 2.22
NWT - 86C,F,(B,G)Parts	57810	2.51 +/- 1.07	.47 +/- .43	2.72 +/- 1.31	5.70 +/- 2.72
Newfoundland	1518	1.25 +/- .77	.19 +/- .34	.96 +/- .68	2.40 +/- 1.80
Nova Scotia	144433	1.63 +/- .80	.37 +/- .44	1.36 +/- .64	3.36 +/- 1.66
Prince Edward Island	8560	2.00 +/- .64	.32 +/- .31	1.10 +/- .45*	3.42 +/- 1.41
Quebec	43169	1.56 +/- .75	.23 +/- .38	.89 +/- .62	2.69 +/- 1.67
Ontario	111669	1.77 +/- .82	.28 +/- .37	1.24 +/- .91	3.30 +/- 1.99
Manitoba	96859	2.74 +/- 1.42	.44 +/- .44	2.60 +/- 1.49	5.78 +/- 3.48
Saskatchewan	194850	2.01 +/- 1.16	.31 +/- .40	1.65 +/- .98	3.97 +/- 2.43
Alberta	25985	1.29 +/- 1.04	.29 +/- .38	1.34 +/- .96	2.92 +/- 2.30
Northwest Territories	263403	2.59 +/- 1.18	.56 +/- .51	2.55 +/- 1.39	5.69 +/- 3.20
Canada	890446	2.12 +/- 1.17	.40 +/- .45	1.87 +/- 1.24	4.39 +/- 2.86

*Indicates the standard deviation is over-estimated. (See Text)

N.P. In [illegible]

TABLE 8 - Percentiles of the Summer Outdoor Exposure Rate in µR/hr for each area surveyed
(Uncorrected for Vegetation)

AREA	Percentiles 25	50	75	90	95	99
Nova Scotia - 11K,11F(Part)	2.06	2.73	3.61	4.66	5.59	7.74
Nova Scotia - 21A,21B,20O	2.38	3.61	4.79	5.75	6.30	7.39
Nova Scotia - 11D,E,(11F,21H)Parts	2.63	3.41	4.16	4.94	5.56	6.57
Quebec - 12L	1.11	1.72	2.59	3.64	4.49	6.62
Quebec - 31J	2.38	3.14	3.84	4.66	5.08	6.42
Ontario - 31F	2.38	3.19	3.86	4.63	4.96	6.26
Ontario - 31C	2.41	3.47	4.54	5.52	5.99	7.07
Ontario - 41J	2.69	3.75	5.31	7.64	9.10	11.74
Ontario - 52G,J	1.66	2.49	3.46	4.65	5.65	7.95
Manitoba - 64F	2.57	4.28	5.90	7.25	7.98	9.26
Manitoba - 64J	2.96	5.22	7.26	8.93	10.12	12.79
Manitoba - 64K	3.60	6.44	8.93	10.82	12.18	15.70
Manitoba - 64N	4.34	6.60	9.05	12.20	14.43	19.06
Manitoba - 64O	3.36	5.43	7.13	9.28	10.94	15.98
Saskatchewan - 64E	3.04	4.60	5.97	7.38	8.30	10.34
Saskatchewan - 64L	3.04	4.67	6.30	7.78	8.75	10.85
Saskatchewan - (72F,G,J)Parts	5.33	5.84	6.43	6.80	6.93	7.52
Saskatchewan - 74A	3.36	4.73	5.84	6.88	7.65	8.98
Saskatchewan - 74F	1.15	1.52	1.88	2.48	2.83	3.74
Saskatchewan - 74G	1.19	1.59	1.99	2.82	3.35	4.51
Saskatchewan - 74H	1.88	2.96	4.73	6.06	6.83	8.48
Saskatchewan - 74P,64M	2.50	3.73	4.98	6.52	7.64	10.50
Saskatchewan - 74N,O	2.25	3.80	5.32	6.82	7.91	10.91
NWT - 75D,E,(75L,74M)Parts	3.90	5.67	7.66	10.55	13.50	21.35
NWT - (75L,85I)Parts	3.34	4.85	6.53	9.11	11.73	16.85
NWT - (85J,85I)Parts	3.60	4.94	6.58	8.89	10.37	13.22
NWT - 75M,85P	3.34	4.92	6.97	10.26	12.48	16.29
NWT - 85N(Part),85O	4.06	6.15	8.04	9.77	10.96	13.72
NWT - 86A,H,(B,G)Parts	3.80	5.04	6.36	7.64	8.47	10.30
NWT - 86C,F,(B,G)Parts	4.07	5.58	7.22	8.91	10.13	12.77
Newfoundland	1.30	2.12	3.24	4.42	5.12	7.18
Nova Scotia	2.42	3.28	4.15	5.05	5.74	6.91
Prince Edward Island	2.75	3.50	4.19	4.71	4.89	5.46
Quebec	1.68	2.62	3.57	4.43	4.94	6.49
Ontario	2.15	3.06	4.08	5.36	6.39	9.24
Manitoba	3.27	5.51	7.68	9.94	11.66	16.08
Saskatchewan	2.04	3.90	5.57	6.72	7.52	9.72
Alberta	1.46	2.32	3.84	5.88	7.16	9.58
Northwest Territories	3.74	5.34	7.13	9.30	11.00	15.84
Canada	2.48	3.93	5.74	7.70	9.23	13.30

TABLE 9

Radioelement concentrations of different classes or rocks*

Rock Class	Example	Code	U (ppm) Mean	U (ppm) Range	Th (ppm) Mean	Th (ppm) Range	K (%) Mean	K (%) Range
Acid Extrusives	Rhyolite	--	4.1	1 - 16	11.9	1 - 40	3.1	1 - 6
Acid Intrusives	Granite	(AI)	4.5	0 - 30	25.7	0 - 250	3.4	0 - 8
Basic Extrusives	Basalt	(BE)	0.8	0 - 3	2.2	0 - 9	0.7	0 - 2
Basic Intrusives	Gabbro	(BI)	0.8	0 - 6	2.3	0 - 15	0.8	0 - 3
Ultrabasic	Dunite	--	0.3	0 - 2	1.4	0 - 8	0.3	0 - 1
Chemical Sedimentary Rocks	Gypsum	--	3.6	0 - 27	14.9	0 - 130	0.6	0 - 8
Carbonates	Limestone	(C)	2.0	0 - 18	1.3	0 - 11	0.3	0 - 4
Detrital Sedimentary Rocks	Sandstone	(DS)	4.8	0 - 80	12.4	0 - 360	1.5	0 - 10
Metamorphosed Igneous Rocks	----	--	4.0	0 - 150	14.8	0 - 105	2.5	0 - 6
Metamorphosed Sedimentary Rocks	----	(MS)	3.0	0 - 53	12.0	0 - 90	2.1	0 - 5
Average continental crust			2.7		9.6		2.1	

*Adapted from Table 10C.6 (Killeen, 1979)

TABLE 10

Population data used in computing the average outdoor exposure rate from terrestrial radiation

Province	Population (Units of 10^3)	Exposure Rate (μR/hr)
Newfoundland	541	2.40 +/- 1.80
Nova Scotia	852	3.36 +/- 1.66
Prince Edward Island	124	3.42 +/- 1.41
Quebec	6,299	2.69 +/- 1.67
Ontario	8,264	3.30 +/- 1.99
Manitoba	1,032	5.78 +/- 3.48
Saskatchewan	932	3.97 +/- 2.43
Alberta	1,826	2.92 +/- 2.30
Northwest Territories	46	5.69 +/- 3.20
Population weighted average		3.22 +/- 1.99

N.B. These results are uncorrected for the effects of vegetation (see text).

TABLE 11

Average monthly soil moisture values in southern Ontario

Months	Soil Moisture (Percent Dry Weight)	Number of Measurements
November to February[1]	39.4 +/- 7.1	30
March[2]	49.9 +/- 7.6	24
April	43.1 +/- 8.1	23
May	40.3 +/- 9.4	122
June	33.7 +/- 10.3	196
July	29.8 +/- 10.3	240
August	32.0 +/- 10.3	237
September	30.4 +/- 10.1	131
October	36.9 +/- 9.3	83

[1] Average of November and December values.

[2] Average of highest values of each site (saturated ground).

TABLE 12

Correction factors to be applied to summer outdoor exposure rates to derive annual values .

Effect	Percentage Change
Attenuation by Snow	-20
Seasonal Soil Moisture Changes	- 5
Attenuation of Airborne Signal by Vegetation	+15

Total Reduction = 10 percent

TABLE 13

Average outdoor exposure rates for some countries (UNSCEAR, 1977)

Country	Population (Units of 10^6)	Exposure Rate ($\mu R\ h^{-1}$)
Austria	10	5.8
East Germany	17	10.5
West Germany	62	6.8
India	633	4.1
Italy	56	8.3
Japan	35	4.7
Poland	7	6.7
Switzerland	219	8.5
Taiwan	12	6.9
United States	219	5.2
Population Weighted Average		5.2
Canada*	24	3.7 +/- 2.3

*Population weighted summer value - corrected by 15 percent for the effects of vegetation (see text).

TABLE 14

Annual whole-body dose-equivalents from internal source of radioactivity

Source	Annual Dose Equivalent (mrem)
Potassium	17
Rubidium-87	0.4
Carbon-14	1.3
TOTAL	18.7

1 Sv = 100 rem

TABLE 15

Average annual Canadian whole-body outdoor dose-equivalents from natural sources of radiation

Source	Dose Equivalent (mrem)
Cosmic Rays	32 +/- 3
Terrestrial Radiation*	18 +/- 11
Internal Radioactivity	19
TOTAL	69 +/- 11

1 Sv = 100 rem

*Corrected for the effects of vegetation, snow and soil moisture (see text).

TABLE 16

Estimated average annual Canadian whole-body dose-equivalents from natural sources of radiation

Source	Dose Equivalents (mrem)
Cosmic Rays	29 +/- 3
Terrestrial Radiation	27 +/- 17
Internal Radioactivity	19
TOTAL	75 +/- 17

1 Sv = 100 rem

FIGURES

1. Published radioactivity profile from Nova Scotia.
2. An airborne gamma-ray spectrum showing the four energy bands.
3. Potassium count-rate variation with aircraft altitude.
4. Variation of potassium and thorium count-rate over the Breckenridge calibration range.
5. Variation of potassium and uranium count-rate over the Breckenridge calibration range.
6. Comparison of ground and airborne potassium measurements.
7. Comparison of ground and airborne uranium measurements.
8. Comparison of ground and airborne thorium measurements.
9. Location of 5km airborne gamma-ray spectrometric surveys carried out with the GSC system.
10. Distribution of radiation exposure rates for areas in Nova Scotia and Quebec.
11. Distribution of radiation exposure rates for areas in Ontario.
12. Distribution of radiation exposure rates for areas in Manitoba.
13. Distribution of radiation exposure rates for four areas in Saskatchewan.
14. Distribution of radiation exposure rates for five areas in Saskatchewan.
15. Distribution of radiation exposure rates for four areas in the Northwest Territories.
16. Distribution of radiation exposure rates for three areas in the Northwest Territories.

17. Distribution of radiation exposure rates for Newfoundland, Nova Scotia and Prince Edward Island.

18. Distribution of radiation exposure rates for Quebec, Ontario and Manitoba.

19. Distribution of radiation exposure rates for Saskatchewan, Alberta and the Northwest Territories.

20. Distribution of radiation exposure rates for Canada.

21. The mean radiation exposure rate in R/hr for each area surveyed.

22. The relationship between the exposure rate for outcrop and the associated overburden as calculated from their potassium, uranium and thorium concentrations.

23. A simplified geology map of Nova Scotia showing the granites of two different ages.

24. The exposure rate map of Nova Scotia calculated from the ground concentrations of potassium, uranium and thorium.

25. Preliminary compilation of airborne data showing the distribution of uranium.

26. Variation with altitude of the annual dose equivalent from the ionizing component of cosmic radiation.

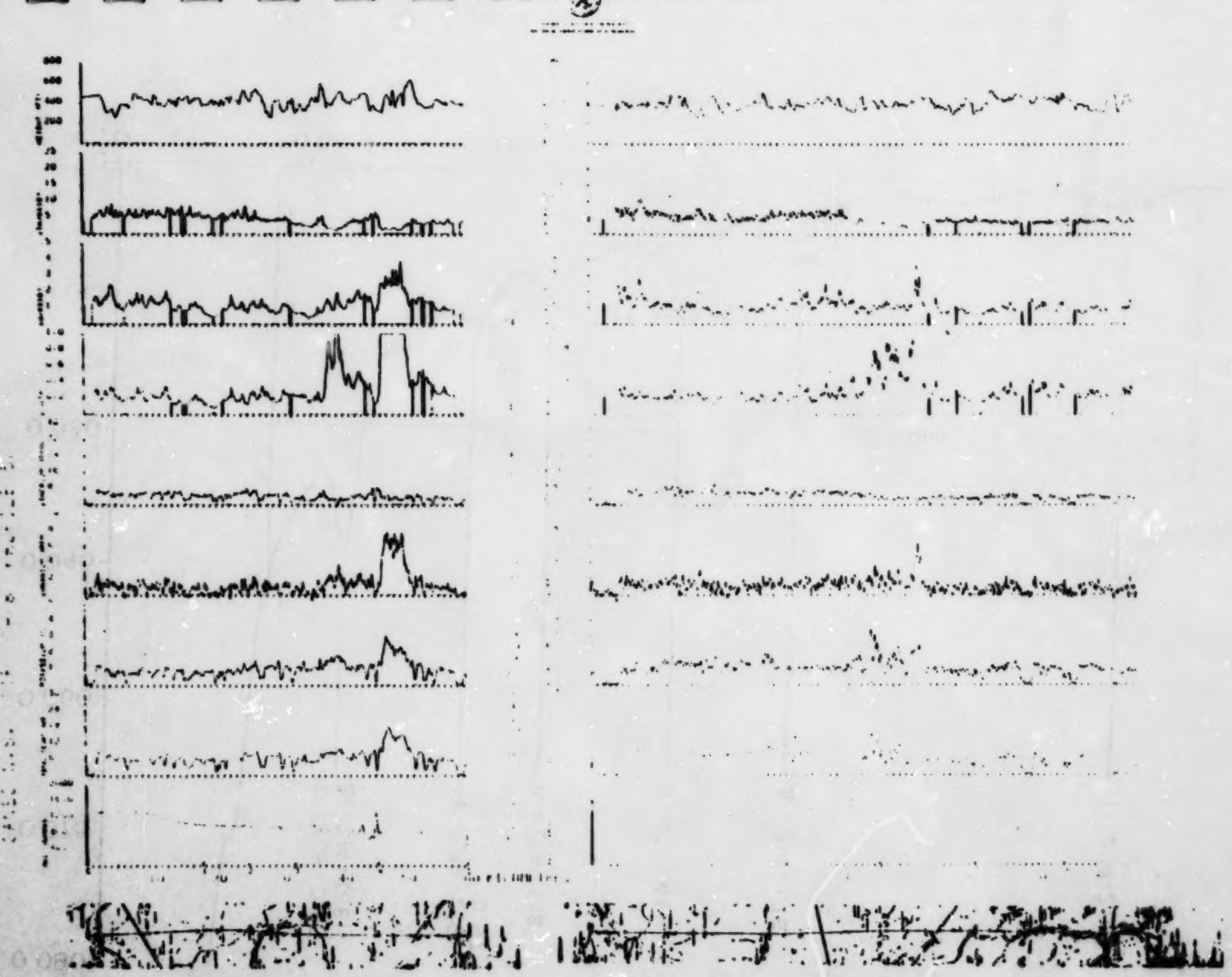


Figure 1 - Published radioactivity profile from Nova Scotia.

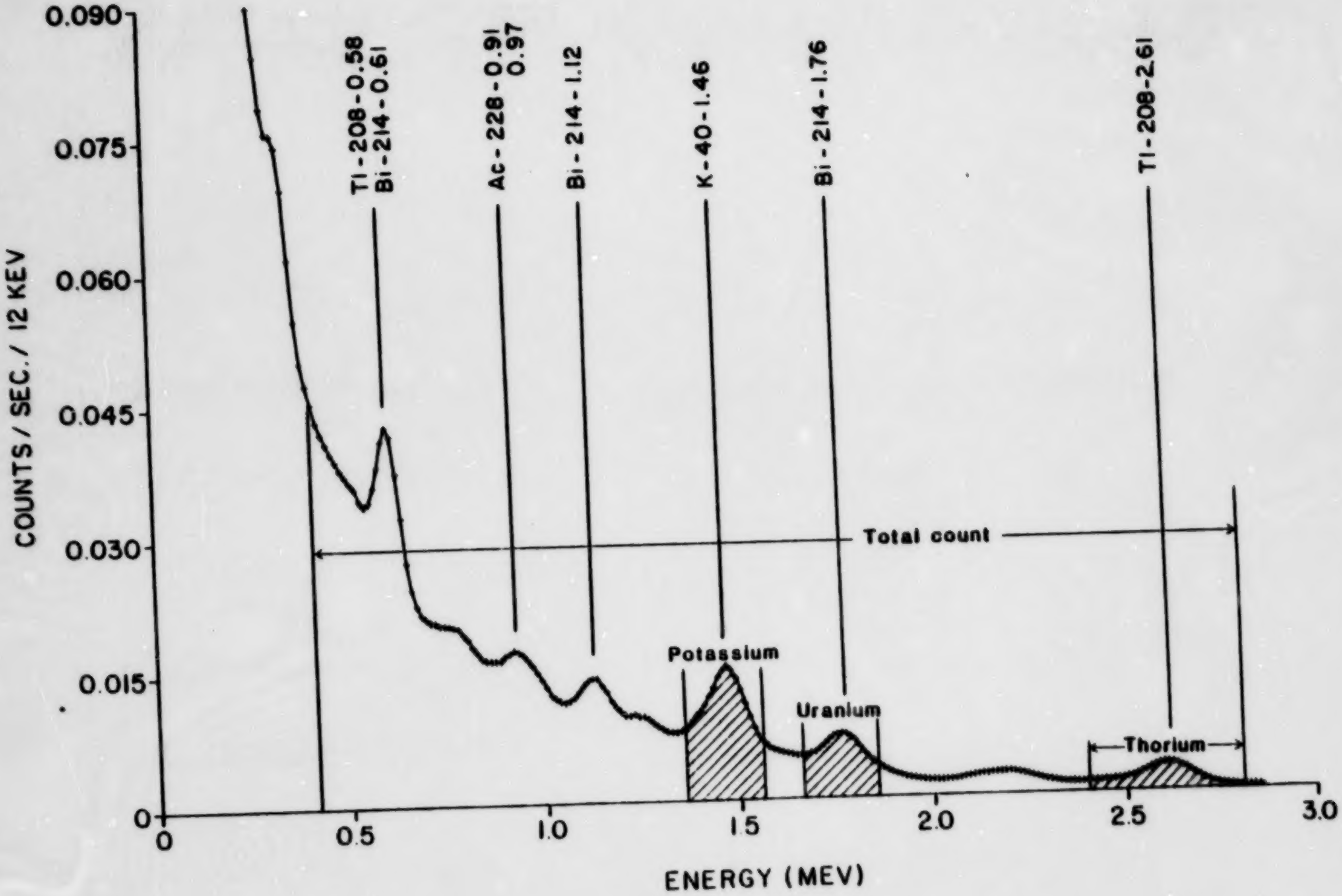
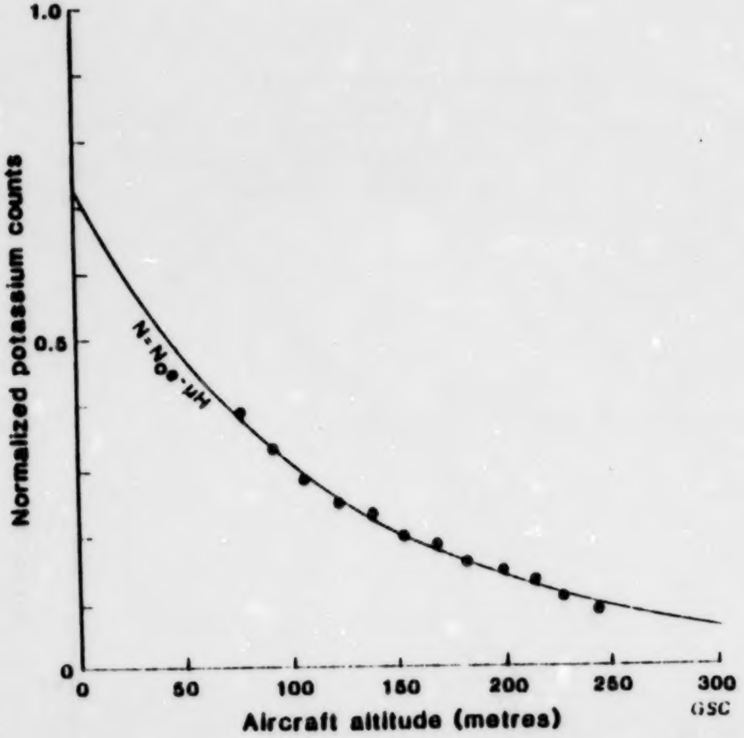


Figure 2 - An airborne gamma-ray spectrum showing the four energy bands.



GSC

Figure 3 - Potassium count-rate variation with aircraft altitude.

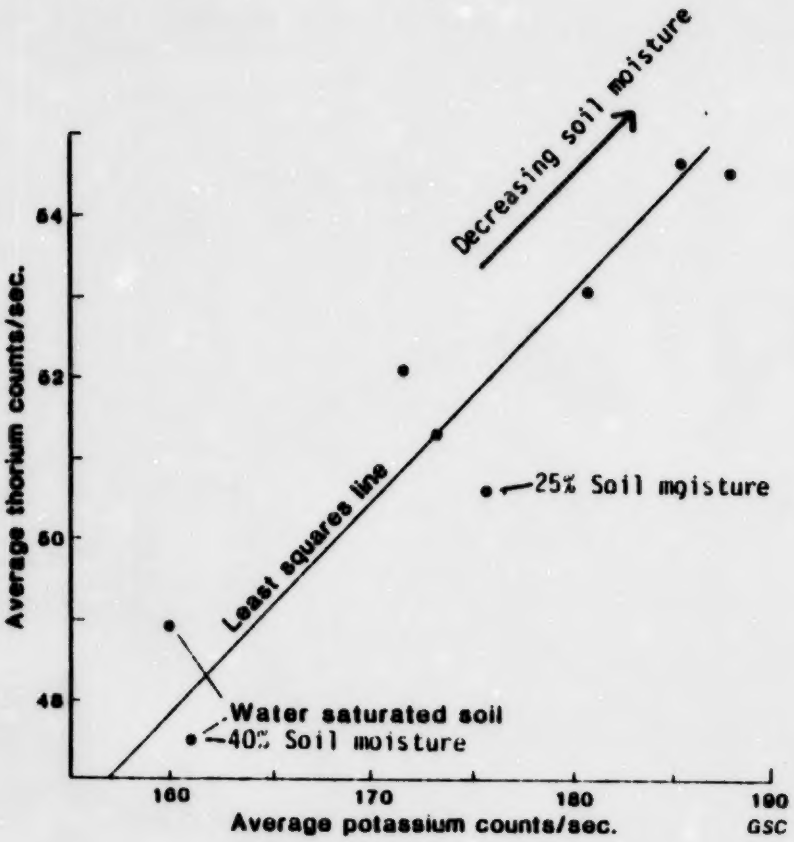


Figure 4 - Variation of potassium and thorium count-rate over the Breckenridge calibration range.

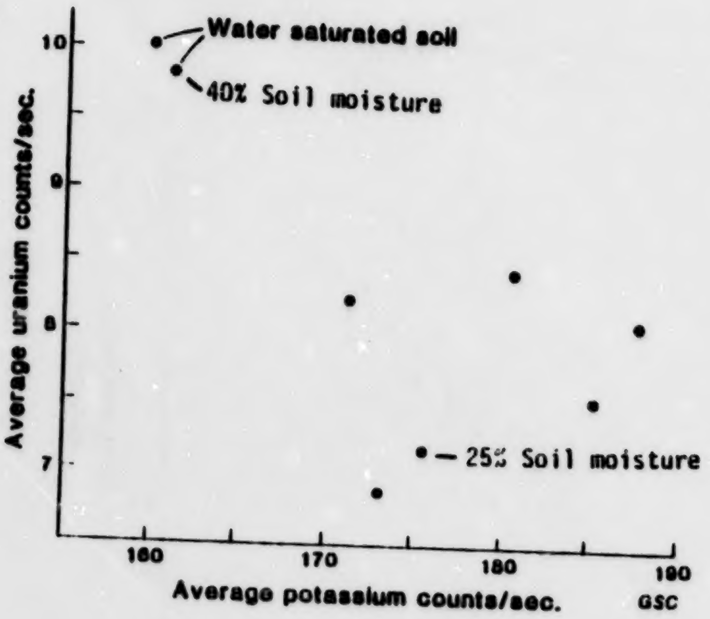


Figure 5 - Variation of potassium and uranium count-rate over the Breckenridge calibration range.

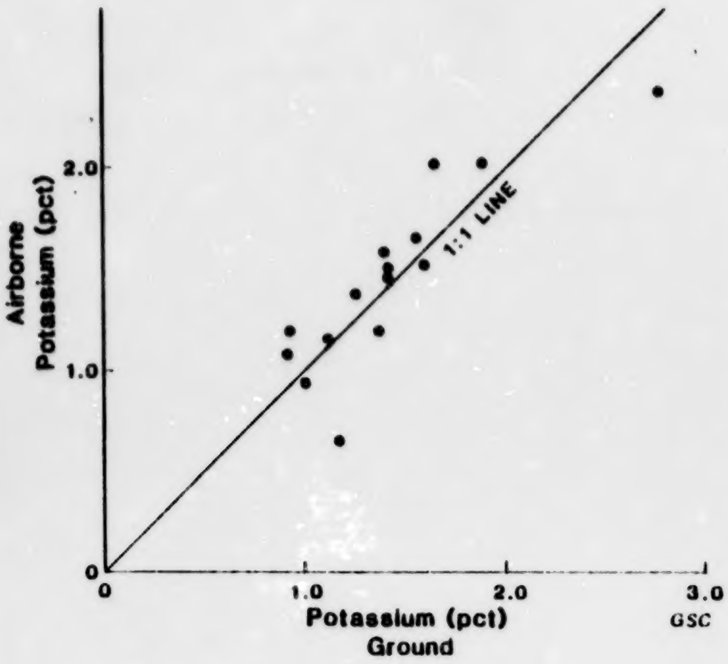


Figure 6 - Comparison of ground and airborne potassium measurements.

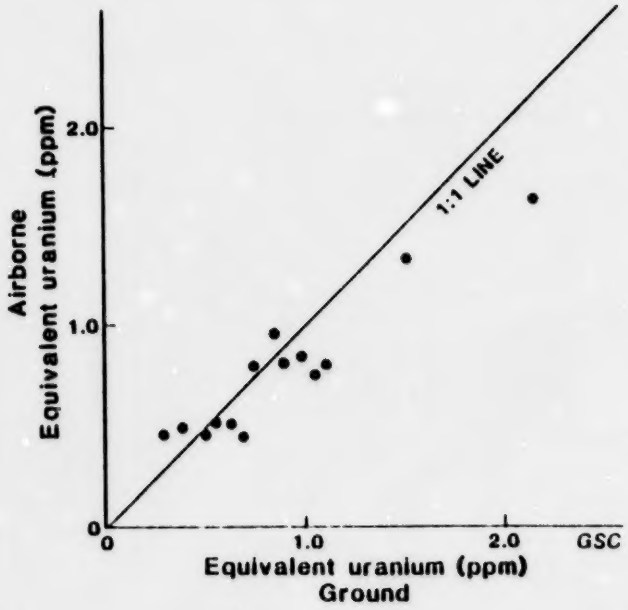


Figure 7 - Comparison of ground and airborne uranium measurements.

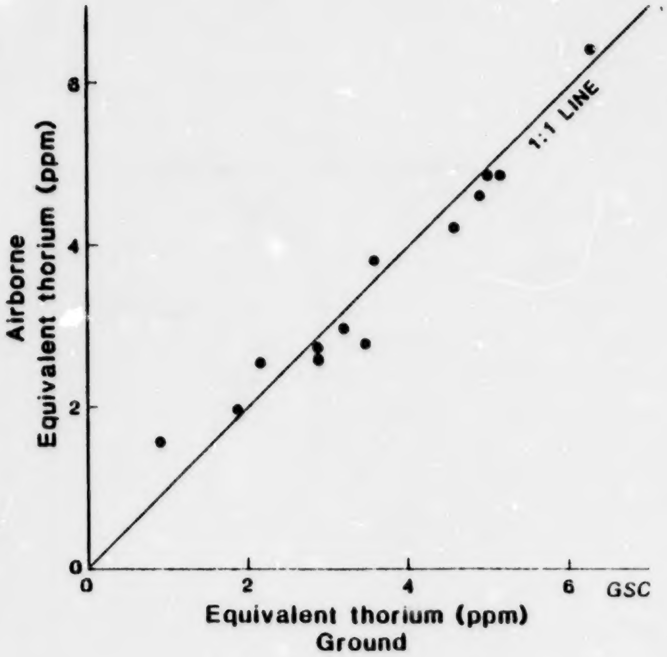


Figure 8 - Comparison of ground and airborne thorium measurements.

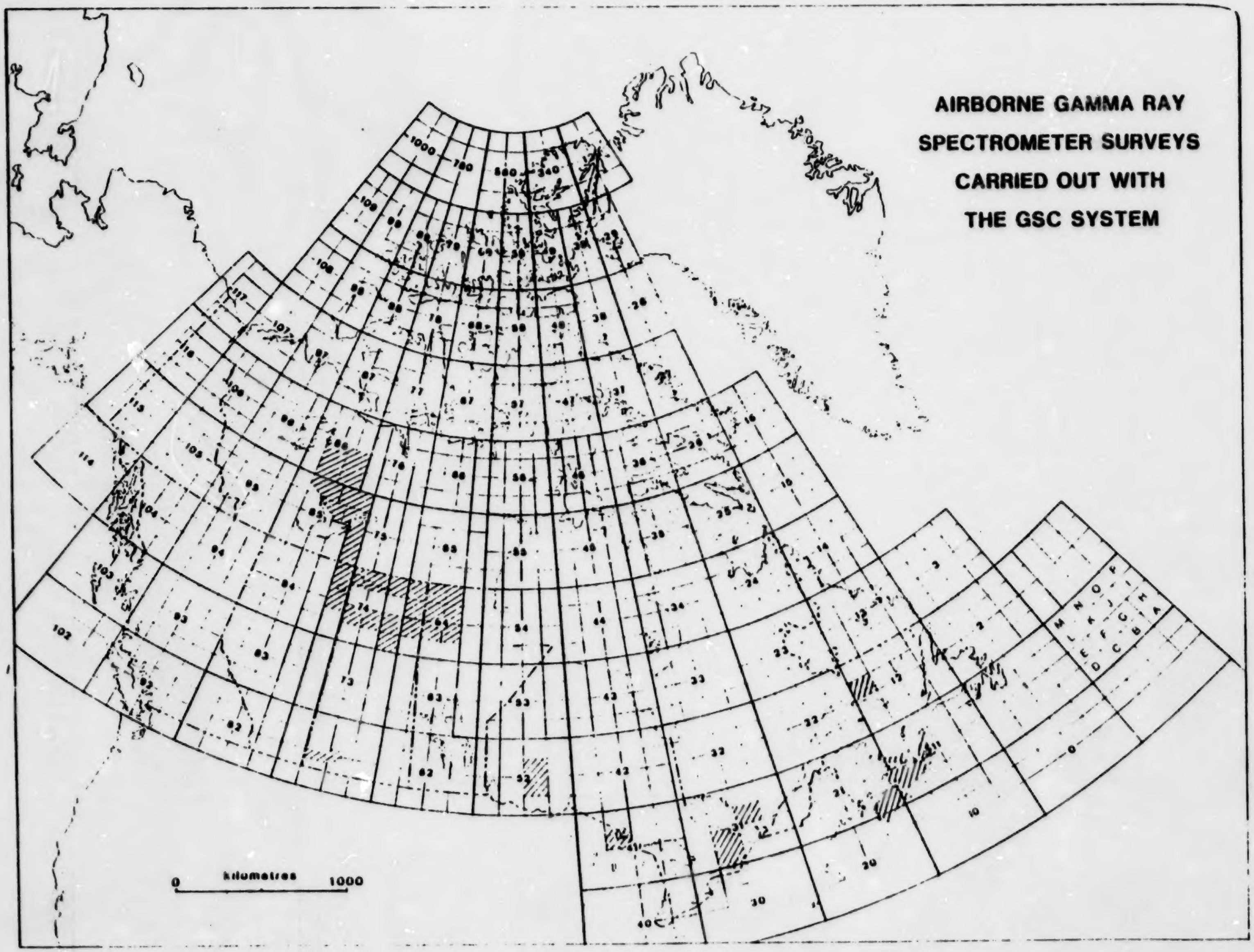

AIRBORNE GAMMA RAY
SPECTROMETER SURVEYS
CARRIED OUT WITH
THE GSC SYSTEM
0
kilometres
1000

Figure 9 - Location of 5 km airborne gamma-ray spectrometric surveys carried out with the GSC system.

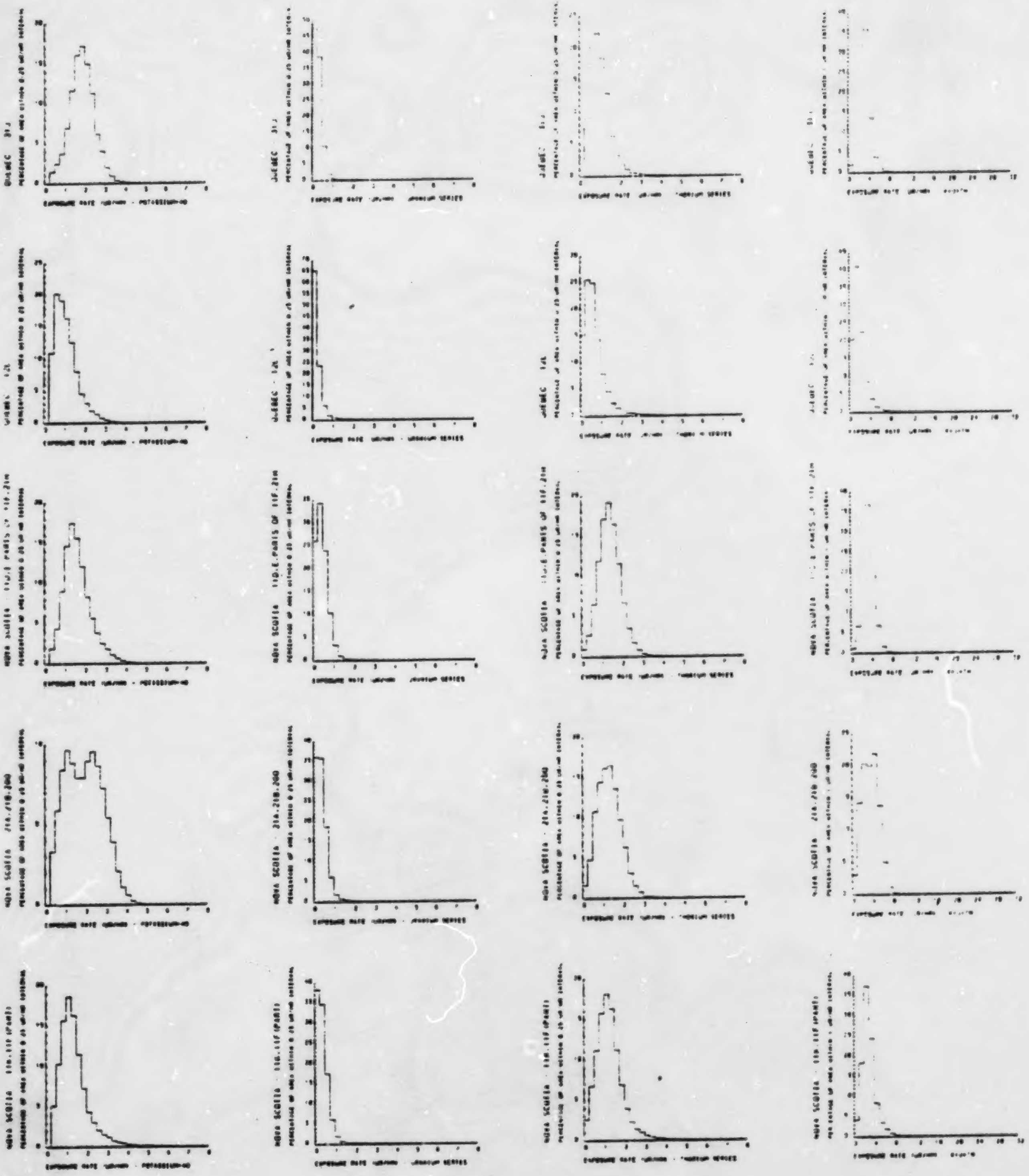


Figure 10 - Distribution of radiation exposure rates for areas in Nova Scotia and Quebec.

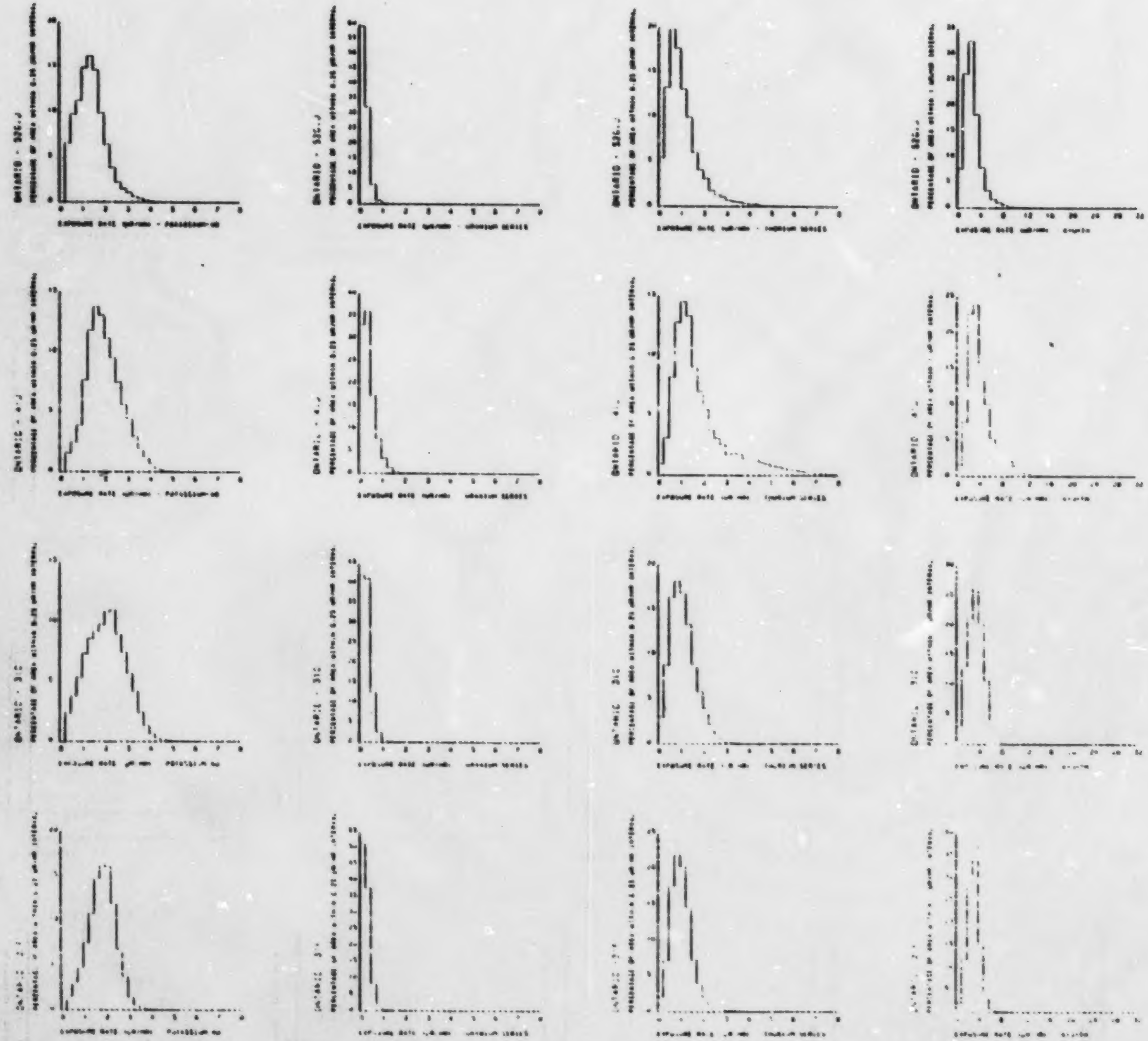


Figure 11 - Distribution of radiation exposure rates for areas in Ontario.

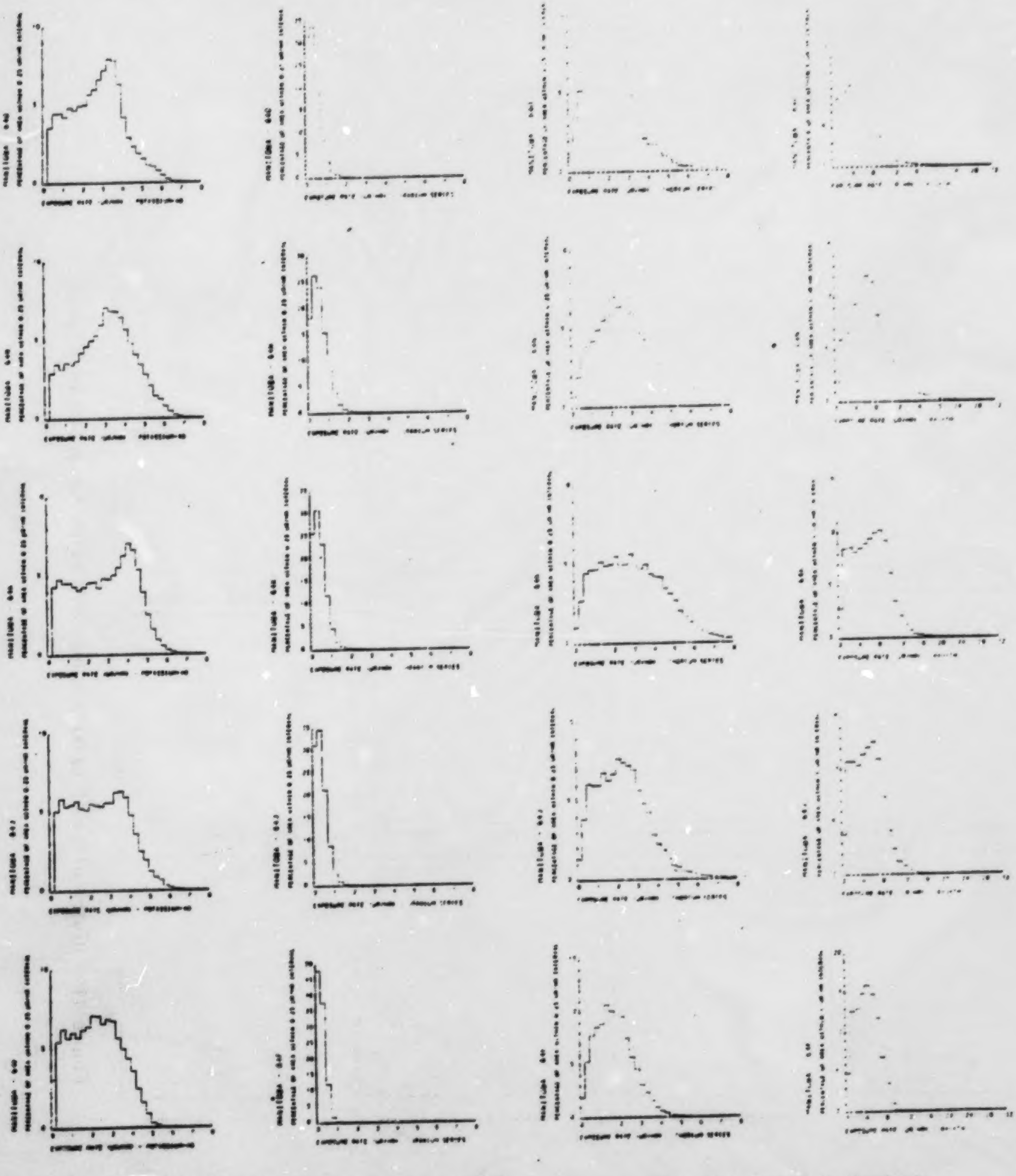


Figure 12 - Distribution of radiation exposure rates for areas in Manitoba.

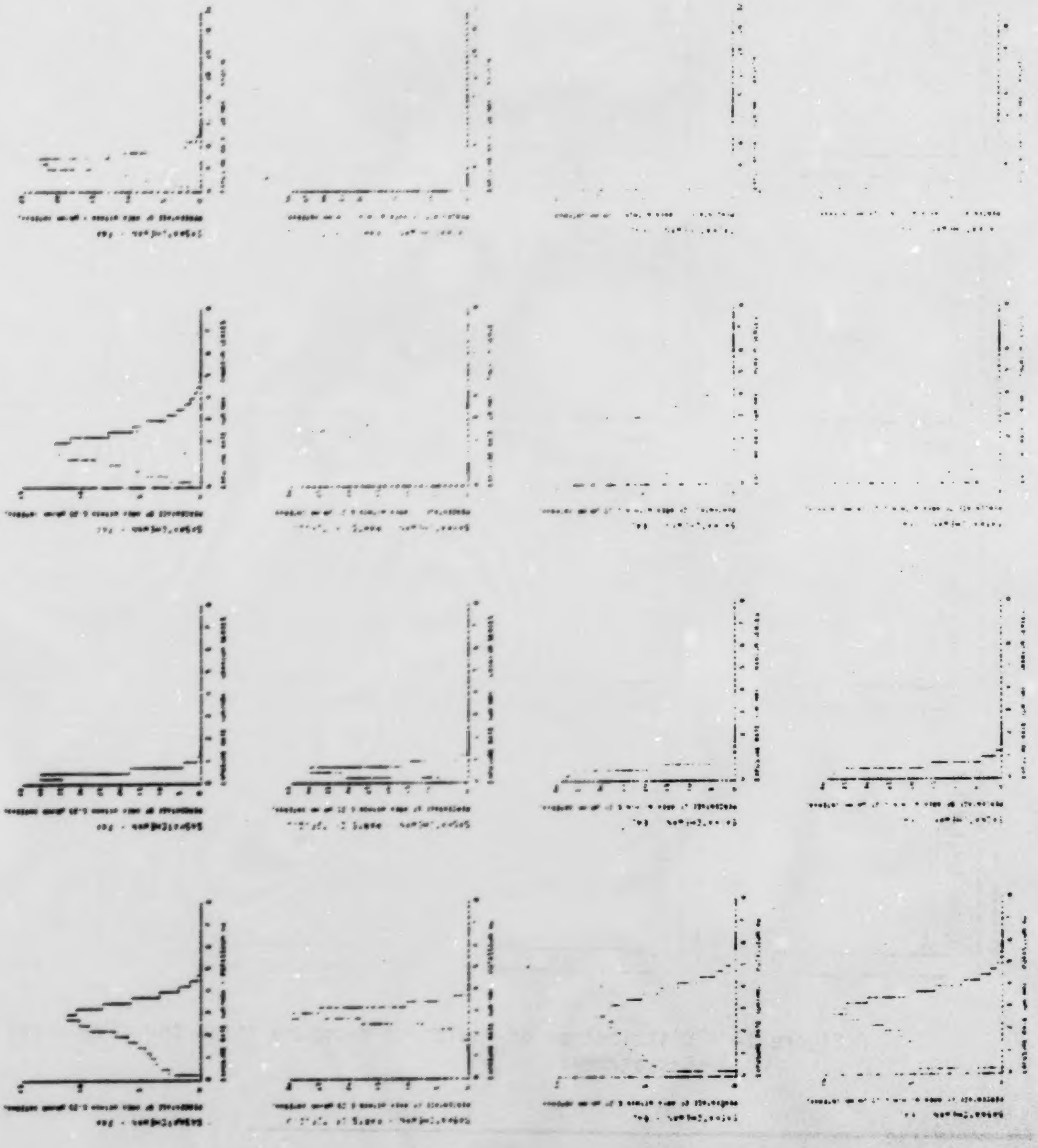


Figure 13 - Distribution of radiation exposure rates for four areas in Saskatchewan.

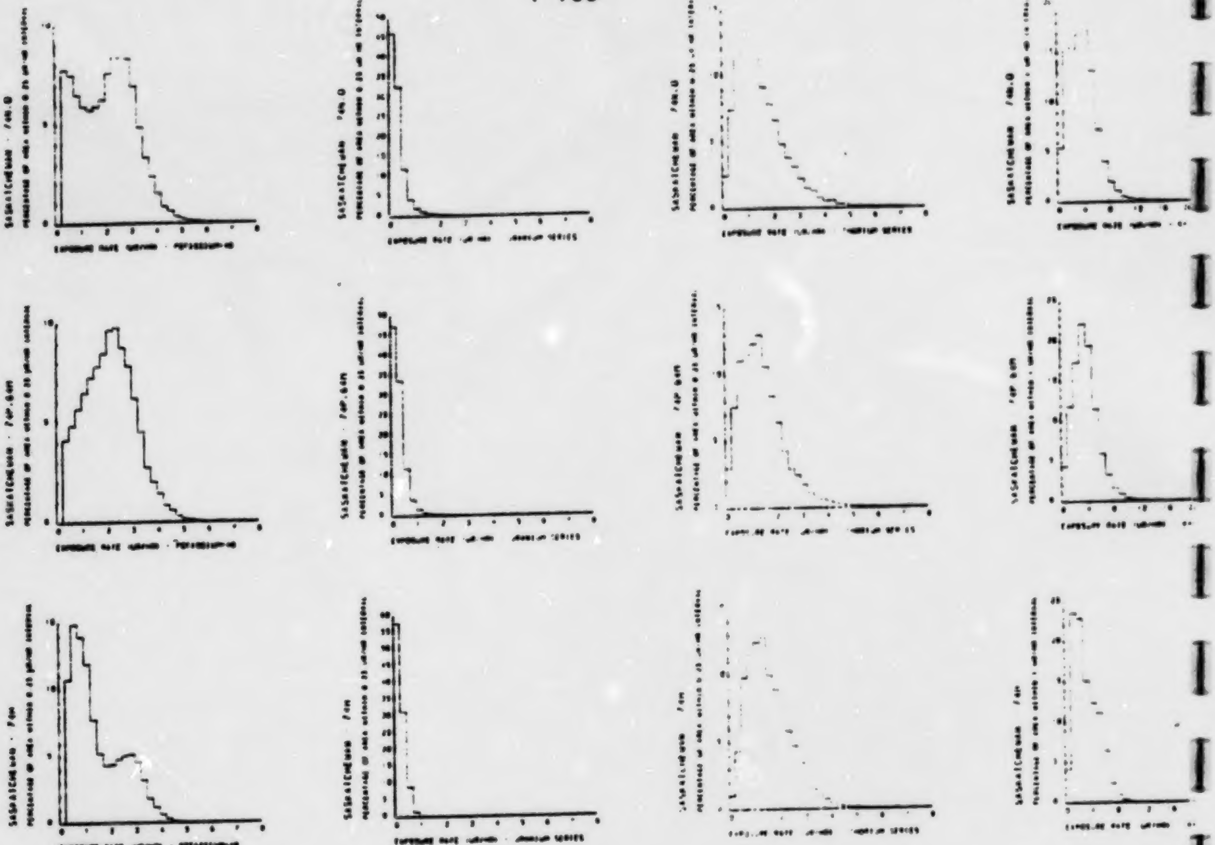


Figure 14 - Distribution of radiation exposure rates for five areas in Saskatchewan.

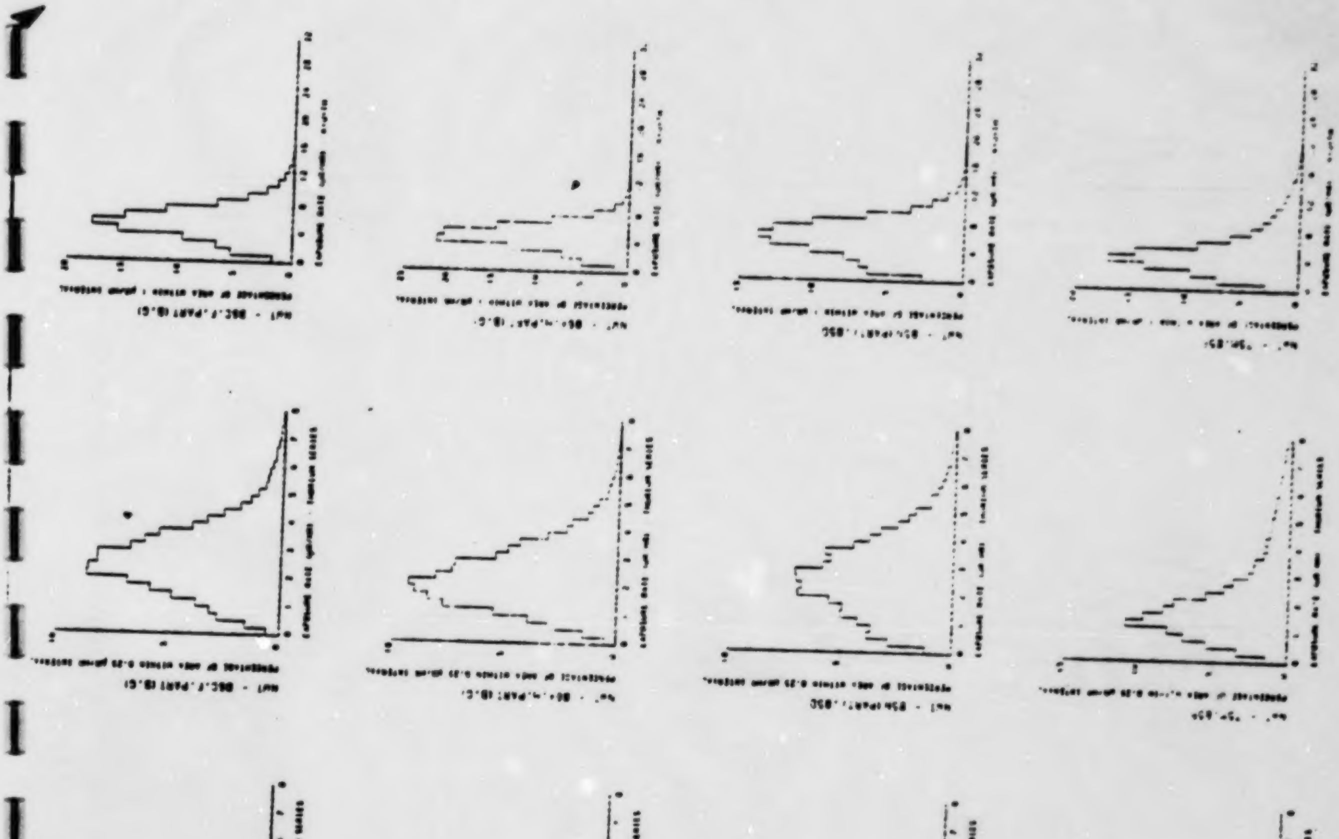


Figure 15 - Distribution of radiation exposure rates for four areas in the Northwest Territories.

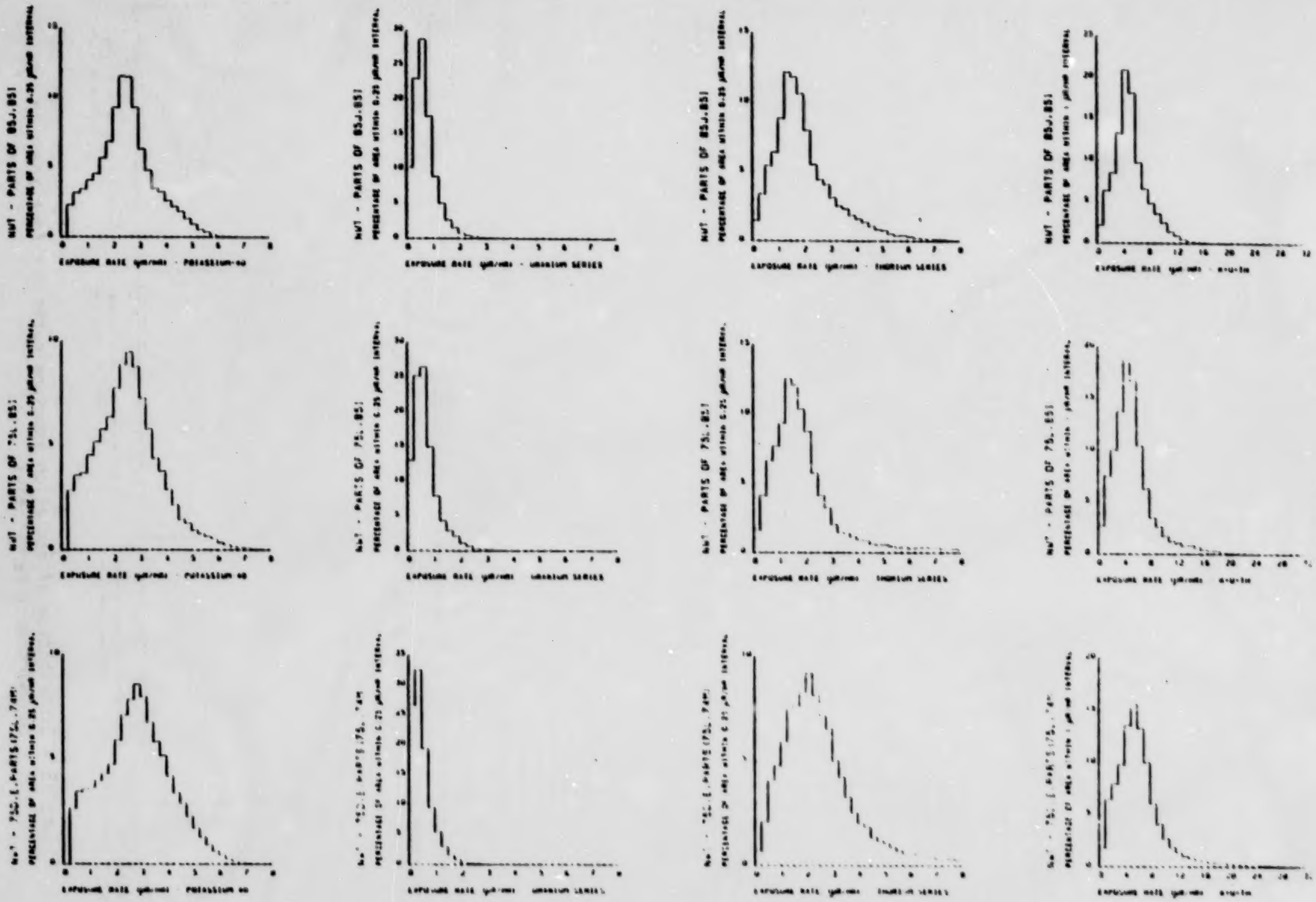


Figure 16 - Distribution of radiation exposure rates for three areas in the Northwest Territories.

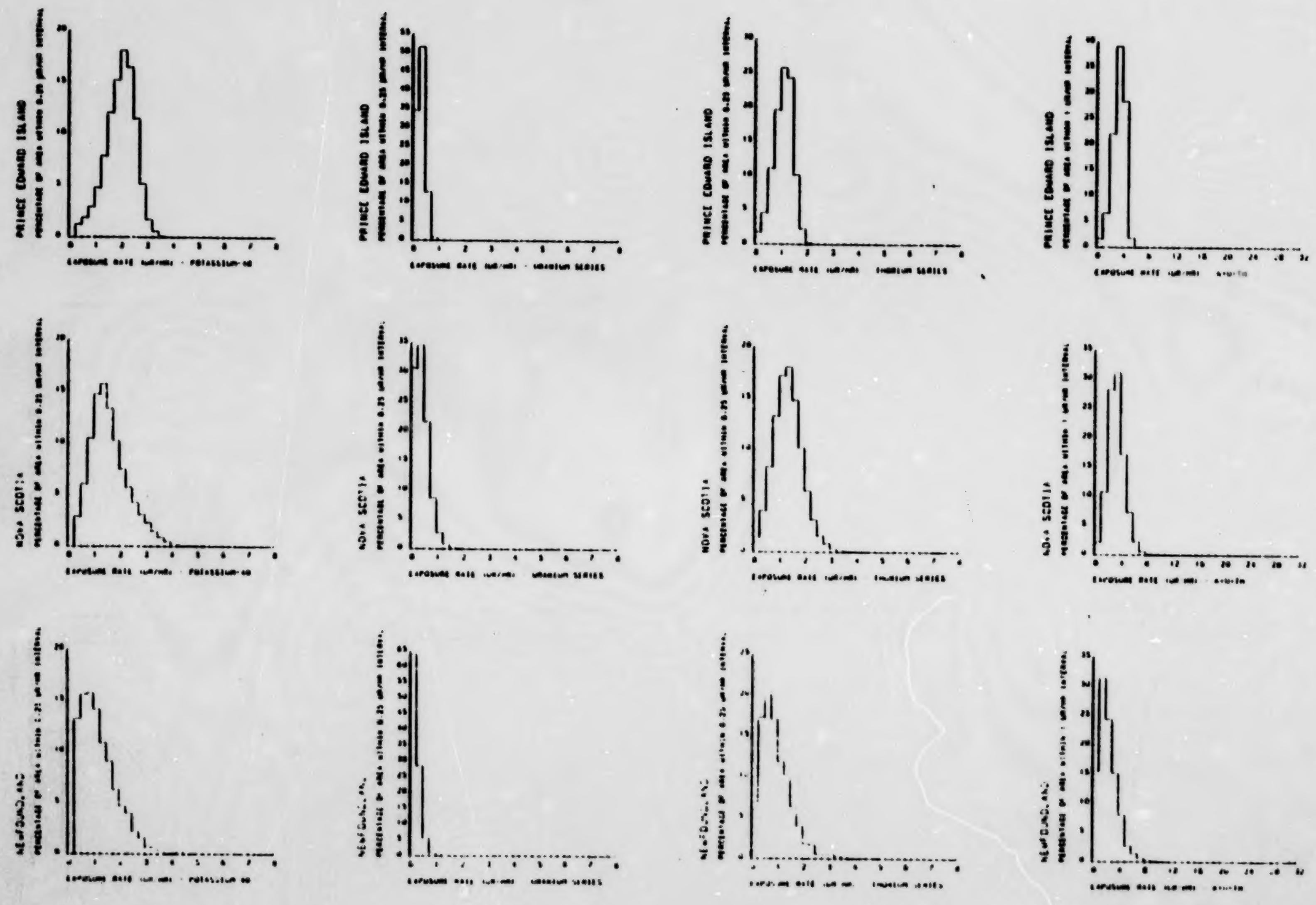


Figure 17 - Distribution of radiation exposure rates for Newfoundland, Nova Scotia and Prince Edward Island.

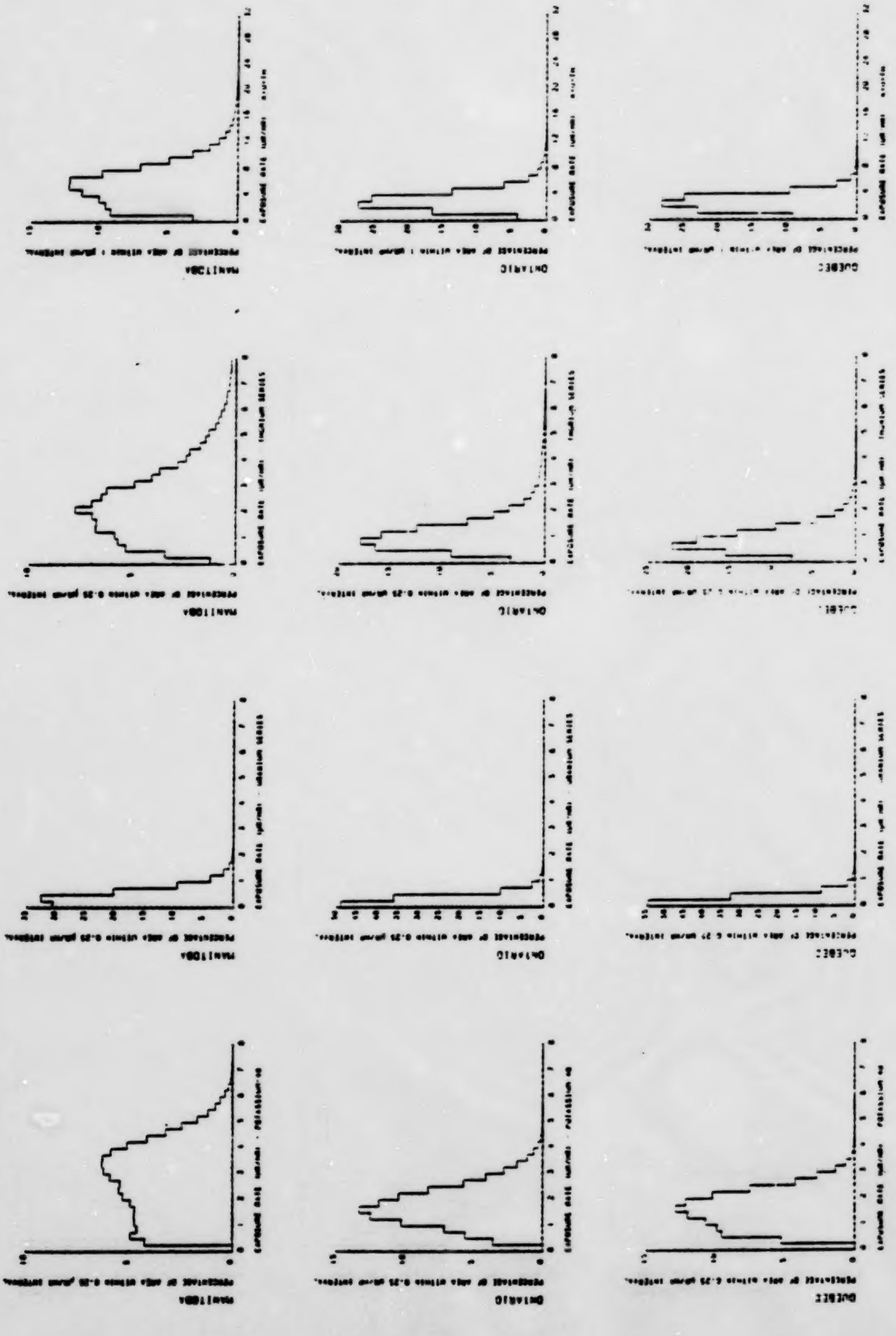


Figure 18 - Distribution of radiation exposure rates for Quebec, Ontario and Manitoba.

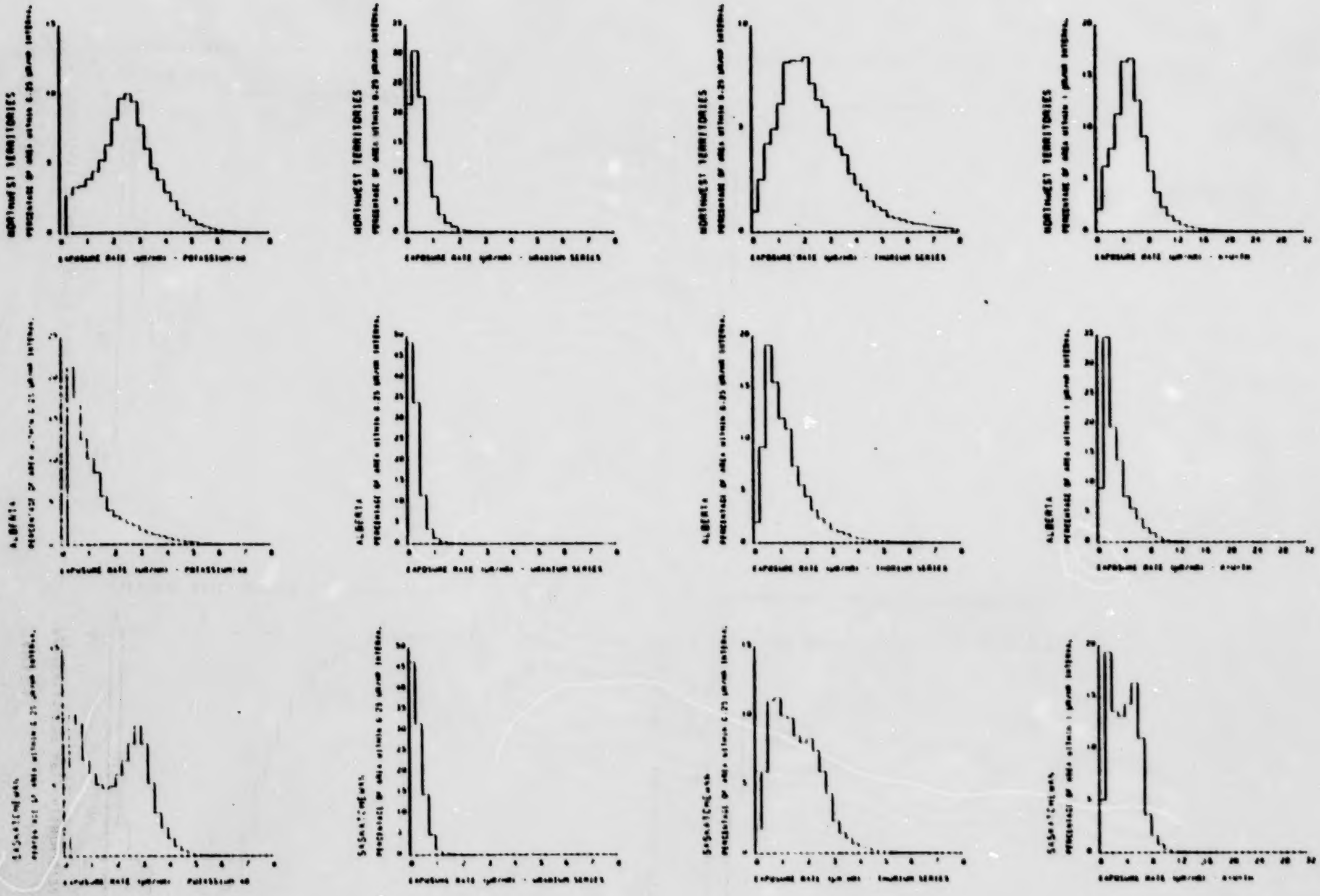


Figure 19 - Distribution of radiation exposure rates for Saskatchewan, Alberta and Northwest Territories.

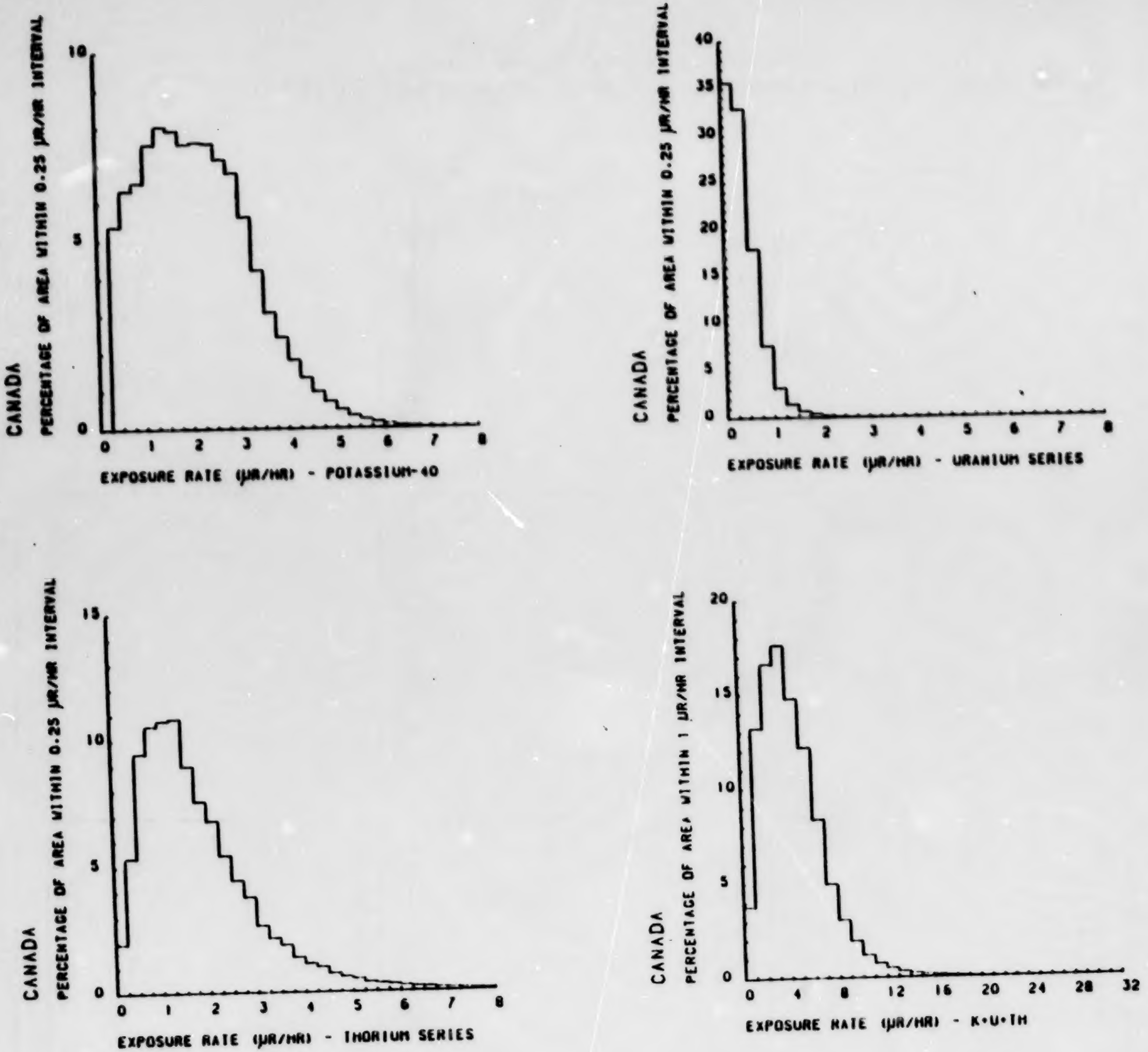


Figure 20 - Distribution of radiation exposure rates for Canada.

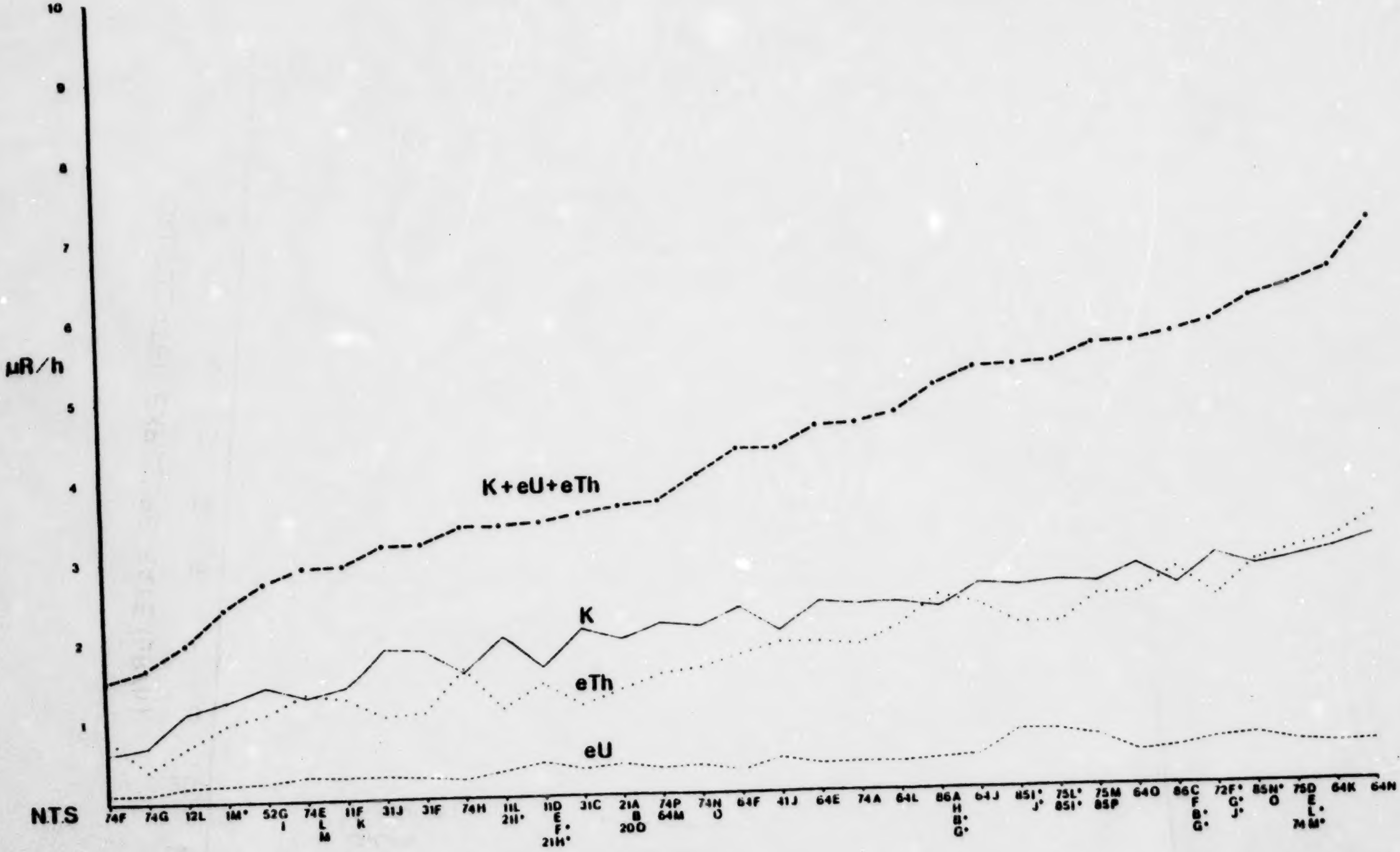


Figure 21 - The mean radiation exposure rate in µR/hr for each area surveyed.

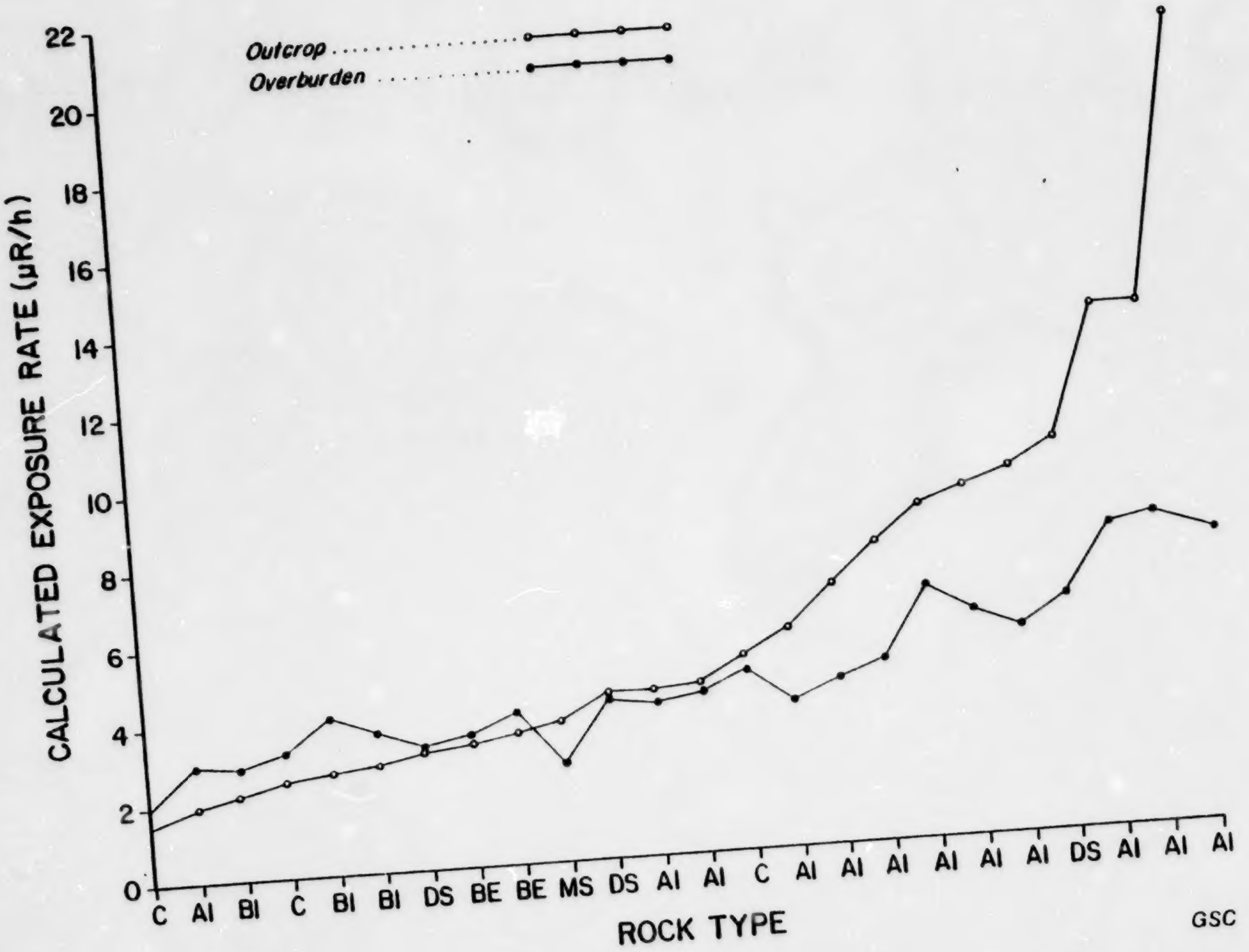


Figure 22 - The relationship between the exposure rate for outcrop and the associated overburden as calculated from their potassium, uranium and thorium concentrations.

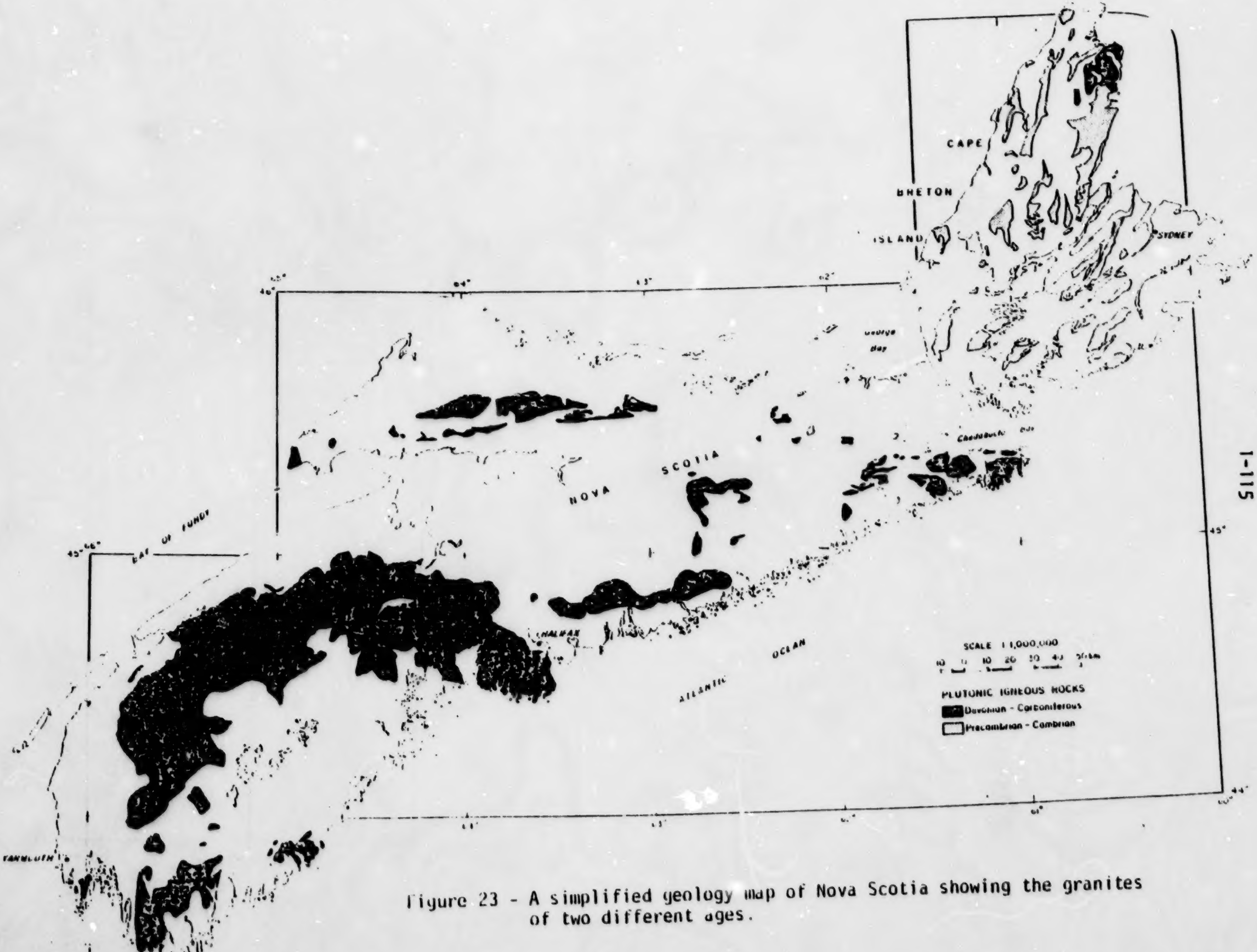

CAPE
BRETON
ISLAND
SYDNEY
George Bay
Chedabucto Bay
NOVA
SCOTIA
BAY OF FUNDY
HALIFAX
ATLANTIC
OCEAN
YARMOUTH
SCALE 1:1,000,000
10 0 10 20 30 40 50 km
PLUTONIC IGNEOUS ROCKS
Devonian - Carboniferous
Precambrian - Cambrian
45°
44°
64°
63°
62°
61°
60°

Figure 23 - A simplified geology map of Nova Scotia showing the granites of two different ages.

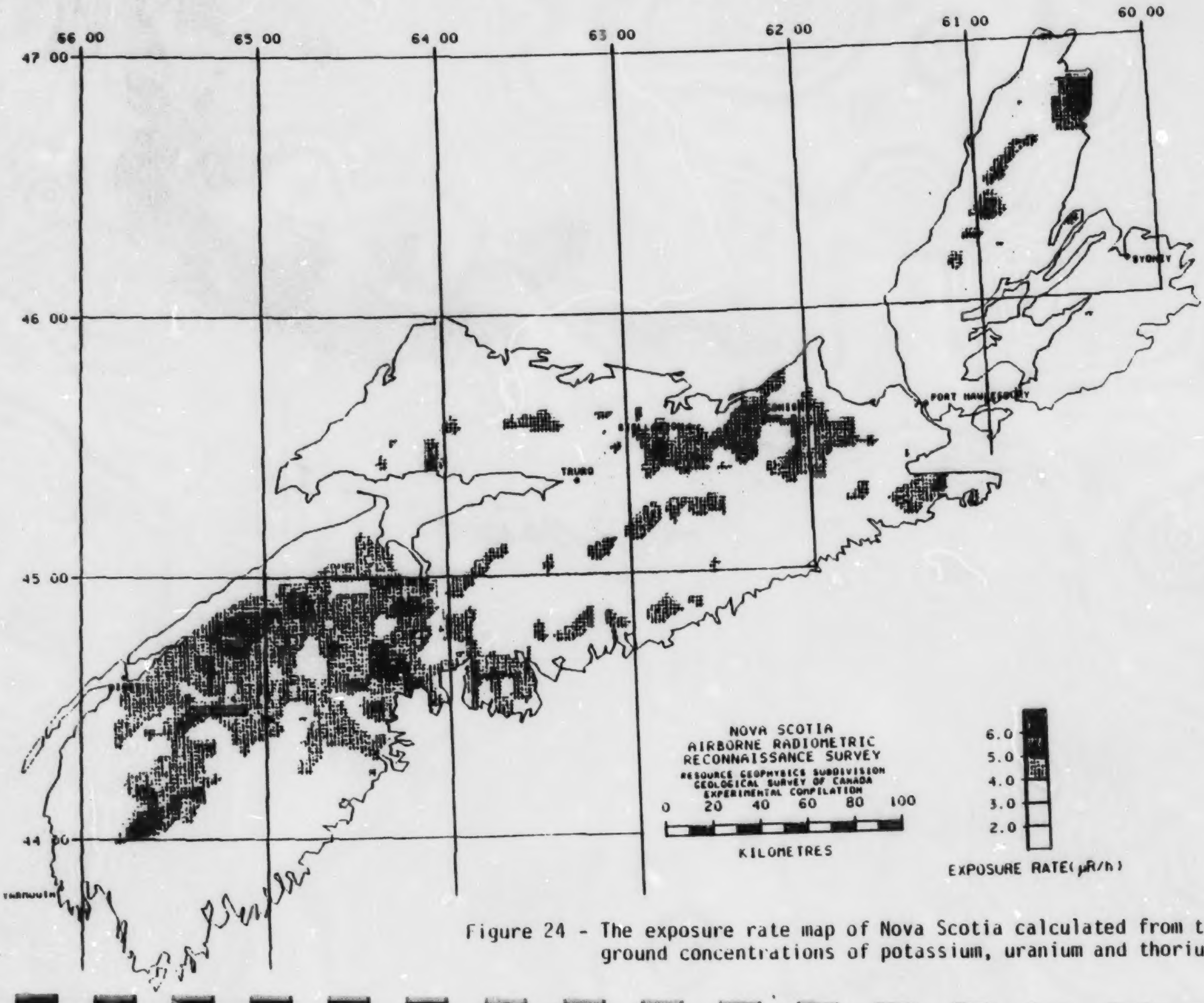

66 00
65 00
64 00
63 00
62 00
61 00
60 00
47 00
46 00
45 00
44 00
SYDNEY
PORT HAWKESBURY
TRURO
YARMOUTH
NOVA SCOTIA
AIRBORNE RADIOMETRIC
RECONNAISSANCE SURVEY
RESOURCE GEOPHYSICS SUBDIVISION
GEOLOGICAL SURVEY OF CANADA
EXPERIMENTAL COMPILATION
0
20
40
60
80
100
KILOMETRES
6.0
5.0
4.0
3.0
2.0
EXPOSURE RATE(μR/h)

Figure 24 - The exposure rate map of Nova Scotia calculated from the ground concentrations of potassium, uranium and thorium.

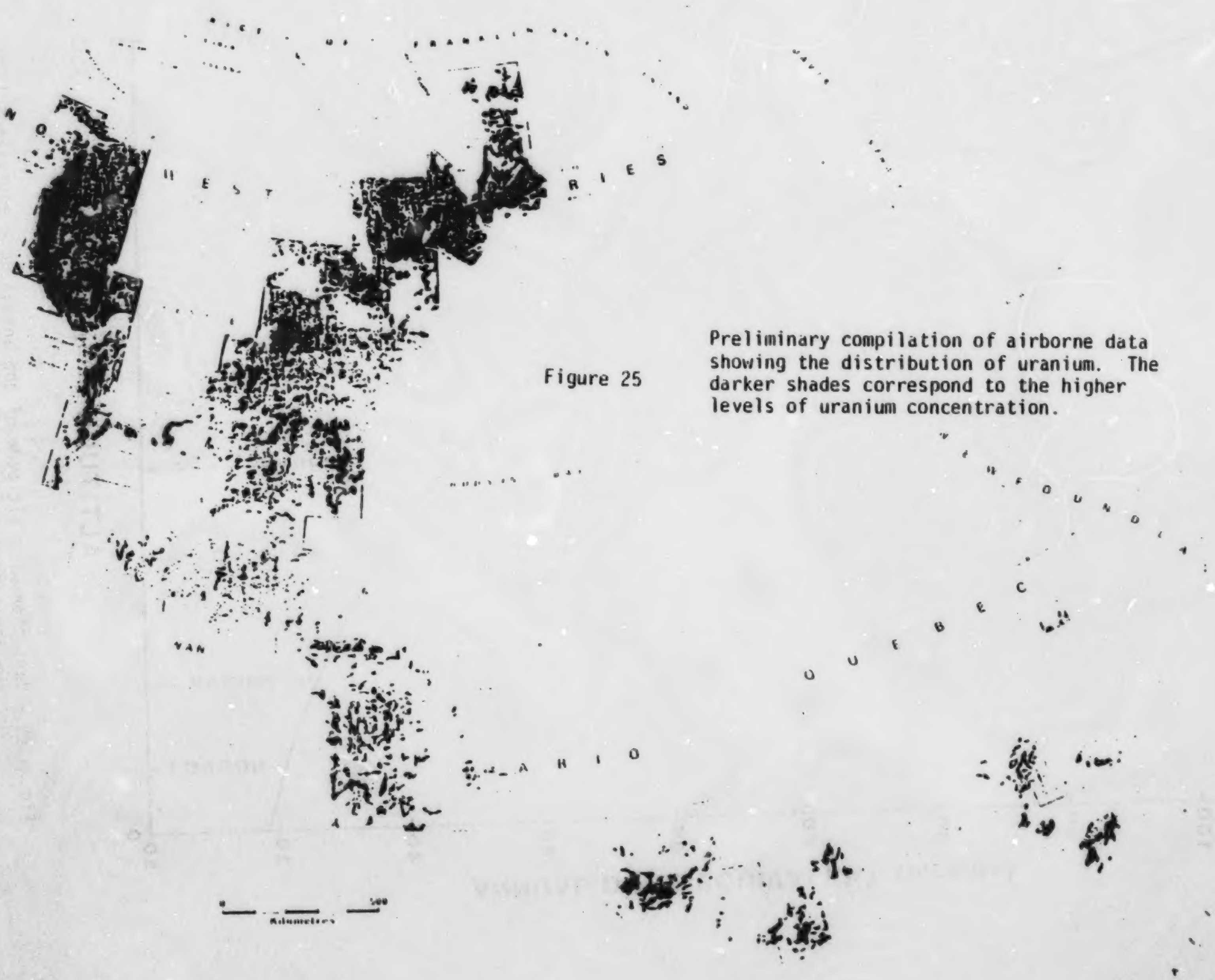
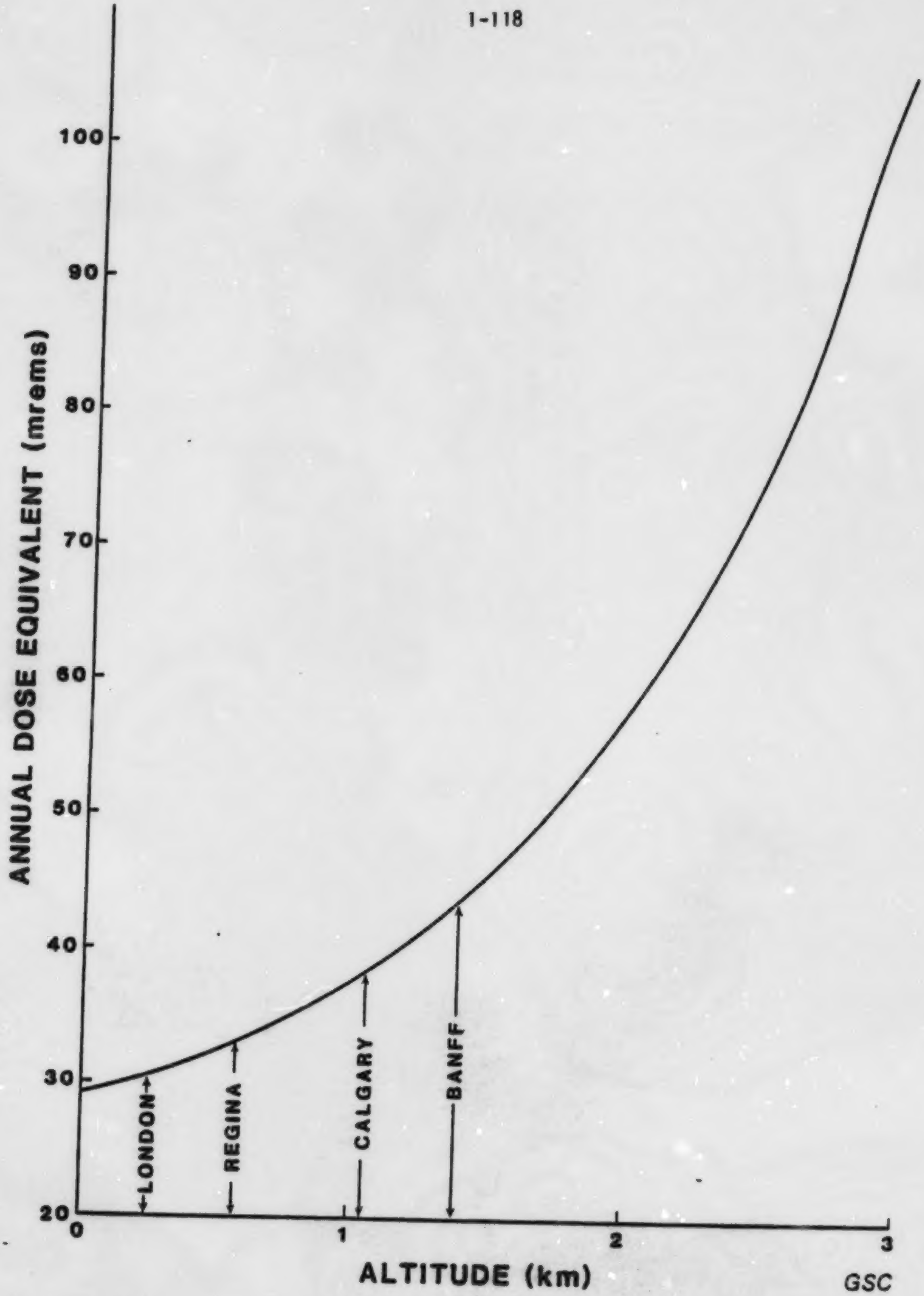

N O
W E S T
R I E S
Q U E B E C
N F O U N D L A
A R I O
0
500
Kilometres

Figure 25

Preliminary compilation of airborne data showing the distribution of uranium. The darker shades correspond to the higher levels of uranium concentration.



GSC

Figure 26 - Variation with altitude of the annual dose equivalent from the ionizing component of cosmic radiation. 1 mrem = 10 μSv

Discussion of Paper by R. Grasty

L. Melis	Can you comment on the radiation exposure of K vs Th?
R. Grasty	They are identical as regards to gamma radiation. The alpha are more damaging. The hazards of both K and Th are the same.
E. Joe	The sandstones are low -- how then were they discovered?
R. Grasty	The occurrence is in very localized seams.
L. Bates	What is the level over the potash in Saskatchewan?
R. Grasty	The levels have not been measured there as yet.
V. Haw	Is the Radium vs the uranium in the tailings easily detected?
R. Grasty	All the Elliot Lake tailings have high levels of activity.
T. Goode	Why was the particular grid of 5 km spacing used in flying?
R. Grasty	In regional mapping it is to determine where to do further investigating. In 1 km survey in Nova Scotia, mapping of the granites have proven very valuable. The amount of funds available for surveying will often dictate the grid.

Canadian Uranium Deposits and Resources

V. Ruzicka
Geological Survey of Canada

Abstract

Canadian uranium deposits occur in various geological environments and contain miscellaneous mineral assemblages: (1) Quartz-pebble conglomerate deposits such as those in the Elliot Lake area, Ontario, occur in lowermost Proterozoic clastic sedimentary rocks and contain uraninite and uranium-titanium aggregates in a heavy mineral assemblage rich in pyrite. (2) Proterozoic unconformity-related deposits such as Key Lake, Rabbit Lake, or Midwest Lake in Saskatchewan are associated with unconformities separating, as a rule, Helikian and Aphebian rocks; in addition to pitchblende, some of these deposits contain coffinite, cobalt and nickel sulphides and arsenides and locally gold. (3) Magmatic/anatectic deposits such as Faraday mine at Bancroft, Ontario, are associated with granitic or syenitic rocks and contain uraninite, uranothorite and allanite along with other accessory minerals, such as magnetite, zircon, sphene and iron sulphides. (4) Vein deposits such as those in the Beaverlodge area, Saskatchewan, or in Millet Brook area, Nova Scotia, occur in metasedimentary, metavolcanic or igneous rocks; their pitchblende, ± coffinite ± brannerite mineralizations occur in fractures and are commonly associated with gangue minerals such as calcite, dolomite, albite, or quartz. (5) The deposits in sandstone or other clastic sediments, such as Blizzard deposit in British Columbia or the Mountain Lake deposit in Northwest Territories, occur as epigenetic concentrations of uranium compounds either in Proterozoic (Mountain Lake) or in Phanerozoic (Blizzard) sedimentary rocks. (6) Volcanogenic deposits, such as Rexspar in British Columbia or Michelin in Labrador, are hosted by trachytic and by rhyolitic rocks respectively; uranium minerals are intimately associated in both of these deposits with titanium minerals.

The major portion of the Reasonably Assured Resources of uranium in Canada occurs in the quartz-pebble conglomerate and Proterozoic unconformity-related deposits. Deposits of the remaining types contain resources either of low grade (e.g. volcanogenic, magmatic/anatectic or sandstone deposits) or irregularly distributed (e.g. vein deposits).

PAPER NOT AVAILABLE AT TIME OF PRINTING.

Discussion of Paper by V. Ruzicka

L. Melis	Do you see any chance of brannerite being found in the Athabasca area?
V. Ruzicka	Yes, in small amounts in almost all the deposits.
N. Aspin	What is the process of accumulation of uranium unconformity in sediments?
V. Ruzicka	It is a low temperature reaction. The deposits were formed after the deposition of the host rocks, Athabasca sand stones, 1.1-1.4 billion years ago. Mineralization may be derived from the basement rocks and moved with water.
J. Robertson	I first want to congratulate you on excellent geology and your presentation. One comment -- although the Roxby Downs deposit is high, the actual uranium production will depend on the copper production.
R. John	The world supply is outside the Communist bloc. What about uranium bearing ores in a country such as Poland?
V. Ruzicka	I really can't comment on that question. Their small production (sandstones) is now shut down.

Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

CANMET
Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

MIGRATION OF CONTAMINANTS FROM UNDERWATER BURIED URANIUM TAILINGS - LABORATORY INVESTIGATIONS OF Ra-226 TRANSFER FLUX

N.K. DAVE AND T.P. LIM

ELLIOT LAKE LABORATORY

MAY 1983

To be presented at CUPMC Meeting, Mississauga, Ontario
May 19-20, 1983.

MINERALS RESEARCH PROGRAM
MINING RESEARCH LABORATORIES
DIVISION REPORT MRP/MRL 83- 47(OP)

MIGRATION OF CONTAMINANTS FROM UNDERWATER BURIED URANIUM TAILINGS - LABORATORY INVESTIGATIONS OF Ra-226 TRANSFER FLUX -

by

N.K. Davé* and T.P. Lim**

ABSTRACT

Laboratory lysimeter tests were performed to study the migration of contaminants from underwater buried pyritic uranium tailings to the overlaying water column. The tests were conducted under two conditions, a continuous flow steady-state condition and a no flow stagnant water column condition. The parameters measured were the mass transfer flux 'J', the transfer coefficient 'K' and the tailings porewater concentration 'C_o' profile as a function of depth and their variations with time for Ra-226 and total dissolved solids (TDS). The results showed that for Ra-226 all the measured parameters decreased with time with the values changing from J = 4.48 to 0.37 $Bq/m^2/hr$ (121 to 10 $pCi/m^2/hr$), K = 0.4 to 0.18 m/month and C_o = 9.9 to 3.9 Bq/L (267 to 105 pCi/L). From the porewater concentration profiles, a molecular diffusional component of the transport flux can be established, but it is small compared to a mass dissolution - transfer component. For total dissolved solids, the measured transfer coefficients K = 0.16 to 0.05 m/month were only one third of those obtained for Ra-226.

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* Research Scientist, ** Physical Scientist, Elliot Lake Laboratory, CANMET, Energy, Mines and Resources Canada, Elliot Lake, Ontario.

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INTRODUCTION

In pyritic uranium tailings, the oxidation of pyrite with the subsequent production of acid and leaching of various metals and radionuclide constituents from the tailings solid phase is a well recognized problem of environmental concern (1-14). In land based tailings management systems, contaminants originating in tailings as a result of pyrite oxidation are transported with precipitation and recharge events as surface run-off and sub-surface seepage to the surrounding environment. Figure 1 shows a typical above grade land based tailings management system where a contaminant plume advances in a nearby aquifer. In such a system the availability of oxygen and moisture control the oxidation rate of pyrite. It is, therefore, considered a viable option to dispose of pyritic tailings underwater thereby limiting the available oxygen and hence minimizing the oxidation and the subsequent leaching processes (15-19). Figure 2 shows an underwater tailings disposal scenario with various components contributing to the transporation of contaminants from tailings to the surrounding environment. Halbert et al. (17) have developed a model for predicting various water quality parameters as a function of time in a water system containing an underwater buried tailings disposal site. In the present investigation, various model input parameters such as flux of dissolved constituents from underwater buried tailings to the overlaying water column and mass transfer coefficients were measured experimentally in a laboratory scale lysimeter. The results obtained for Ra-226 and total dissolved solids are presented in this report. Results for various other parameters will be presented in a separate report.

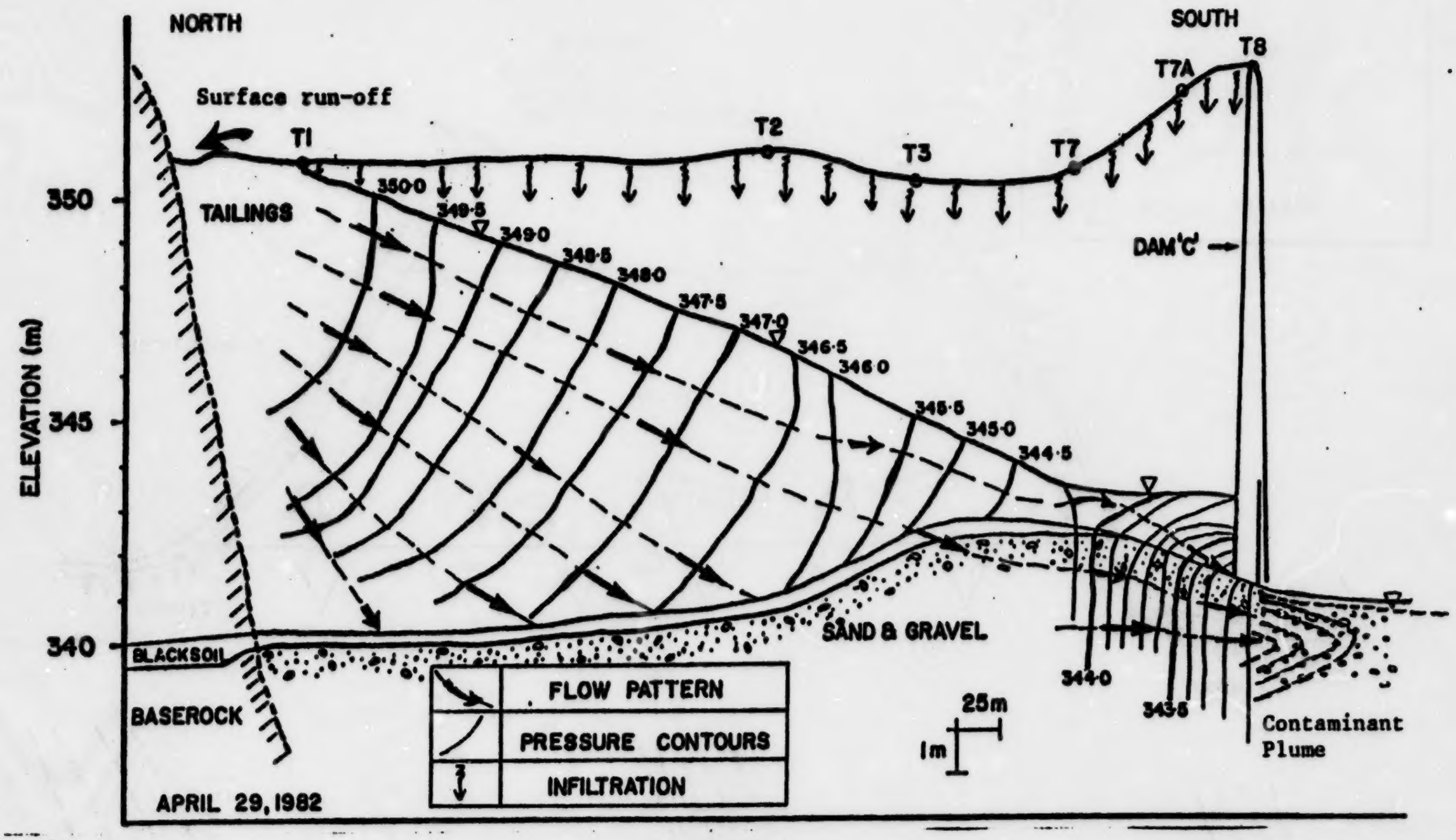
NORTH
SOUTH
Surface run-off
T1
T2
T3
T7
T7A
T8
TAILINGS
DAM 'C'
350·0
349·5
349·0
348·5
348·0
347·5
347·0
346·5
346·0
345·5
345·0
344·5
344·0
343·5
ELEVATION (m)
350
345
340
BLACKSOIL
BASEROCK
SAND & GRAVEL
FLOW PATTERN
PRESSURE CONTOURS
INFILTRATION
25m
1m
Contaminant Plume
APRIL 29, 1982

Fig. 1 - Overland based tailings management scenario - generation of contaminants and their migration as sub-surface seepage and surface runoff.

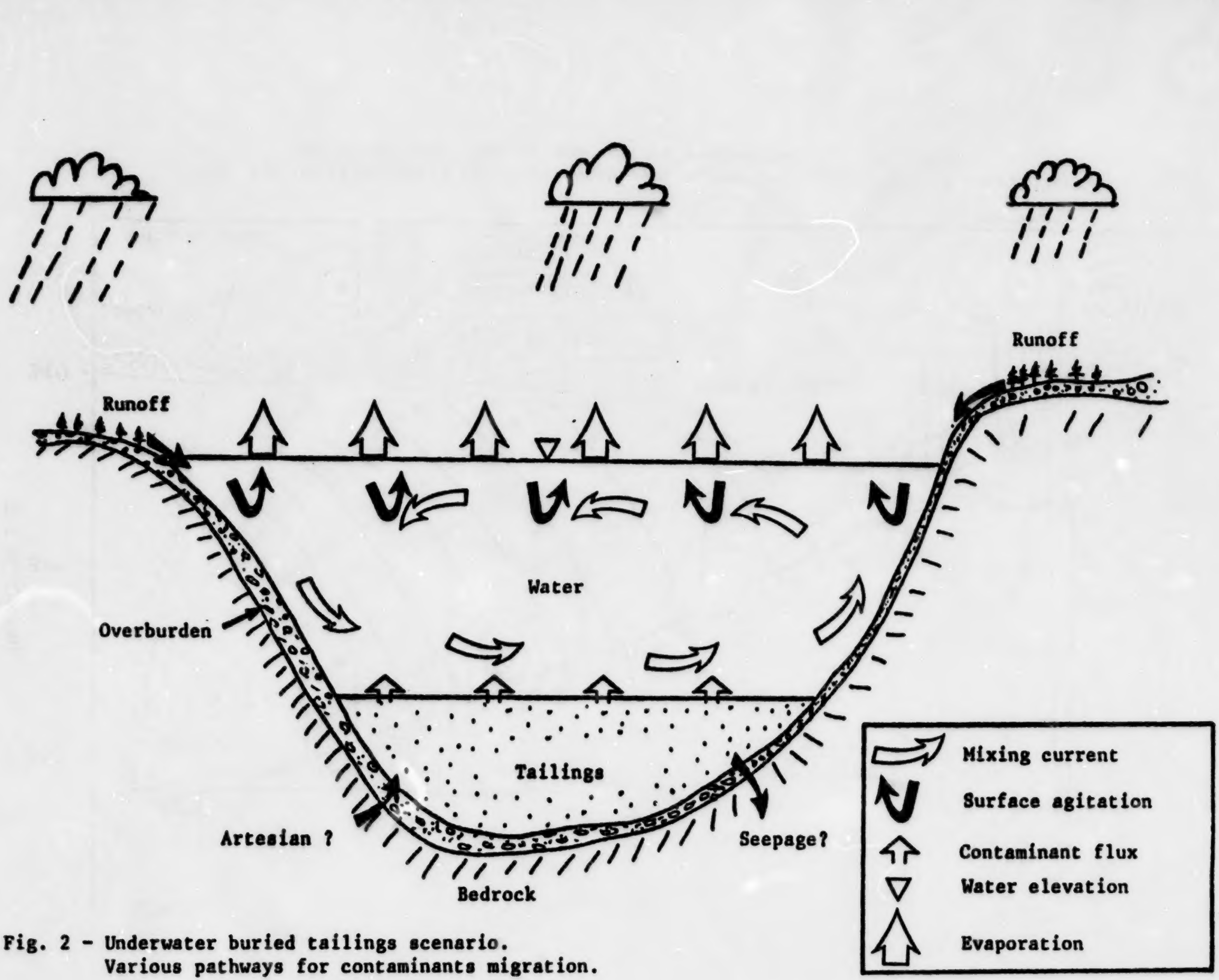
Runoff
Runoff
Water
Overburden
Tailings
Artesian ?
Seepage?
Bedrock
Mixing current
Surface agitation
Contaminant flux
Water elevation
Evaporation

Fig. 2 - Underwater buried tailings scenario.
Various pathways for contaminants migration.

EXPERIMENTAL SET-UP

Figure 3 shows the experimental arrangement for the underwater buried tailings lysimeter. It consisted of a plexiglass chamber 1.22 m long, 0.3 m wide and 0.61 m high. The lower half of the chamber was filled with tailings to a height of 0.3 m and the top half contained the water column as shown in the figure. The height of the water column above the tailings was adjustable with the outflow height control. In order to prevent stratification in the water column, water was continuously homogenized in two mixing side chambers. In the flow experiment, water continuously flowed over the tailings and two baffle plates at the entrance and exit prevented a turbulent flow.

The tailings used in the lysimeter were obtained from Rio Algom's Quirke Mill tailings discharge. The tailings slurry at 20% solids was allowed to settle and the supernatant liquid decanted off as much as possible before transferring to the lysimeter as slurry with a high solid fraction. In the lysimeter, the tailings were allowed to settle up to the desired height and the supernatant liquid was further decanted off. Within the tailings nine mini-piezometers and temperature probes were installed at three different depths in three columns to detect any pressure or temperature gradients. The top, middle and bottom installations were approximately 0.025 m, 0.15 m and 0.275 m below the tailings surface, respectively. The mini-piezometers were narrow diameter CPVC pipes approximately 0.1 m long, perforated and covered with a fibreglass wool screen to prevent siltation. They were placed horizontally along the width of the lysimeter and connected to tygon tubing on the outside. The piezometers also provided tailings porewater sampling ports for various locations.

Experiments were performed under two different conditions. In the steady-state flow condition, a constant flow of water was maintained in the

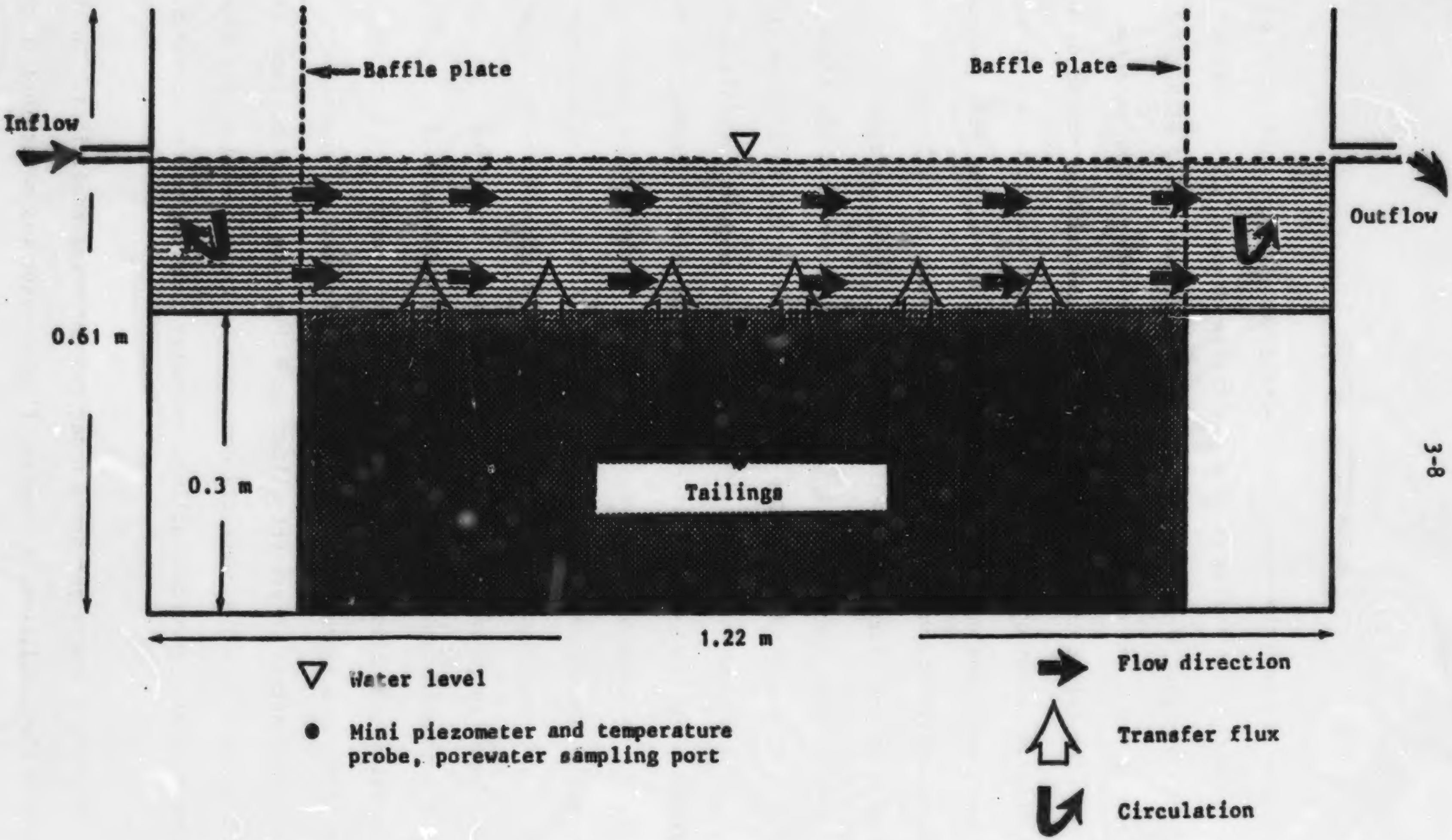
Inflow
Baffle plate
Baffle plate
Outflow
0.61 m
0.3 m
Tailings
1.22 m
▽ Water level
● Mini piezometer and temperature probe, porewater sampling port
Flow direction
Transfer flux
Circulation

Fig. 3 - Underwater buried tailings lysimeter - constant flow configuration.

water column above the tailings. Contaminants flux originating from tailings is transported by the flowing water and thus for a given flow, a constant gradient is maintained along the tailings water interface and a steady-state equilibrium condition is reached. The flow was regulated with a flow regulator and distilled water was used in the experiment at the room temperature to provide low background. During a given flow, the homogeneity of the water column was checked with a conductivity probe. After steady-state, water samples were taken from both the middle of the water column above the tailings and the outflow pipe. The flow was then adjusted to another level and the system was allowed to reach steady-state equilibrium. Measurements were taken for various flows ranging from 2 mL/min to 350 mL/min and with two different elevations of the water column, 0.178 m and 0.055 m, respectively.

In the no-flow condition, the water column above the tailings was first completely flushed, then replaced with a fresh water column of a desired height and the system was closed at time $t = 0$. The water column was then continuously homogenized by mixing with a multi-head peristaltic pump in the mixing chambers and a horizontal cross-flow. This provided minimum disturbance to the tailings surface. Figure 4 shows the experimental arrangement. Water samples from the column were taken at various time intervals and the concentration increase of various constituents as a function of time was measured. Experiments were performed with two water column heights, 0.178 m and 0.055 m, respectively.

Tailings porewater samples were obtained from various sampling locations at different times and all the samples were analyzed for pH, Eh, electrical conductance, total dissolved solids (TDS), NH_4^+, NO_3^-, SO_4^{-2}, total iron, Ca, Al, Mg, Mn, Pb, U, Th, and Ra-226.

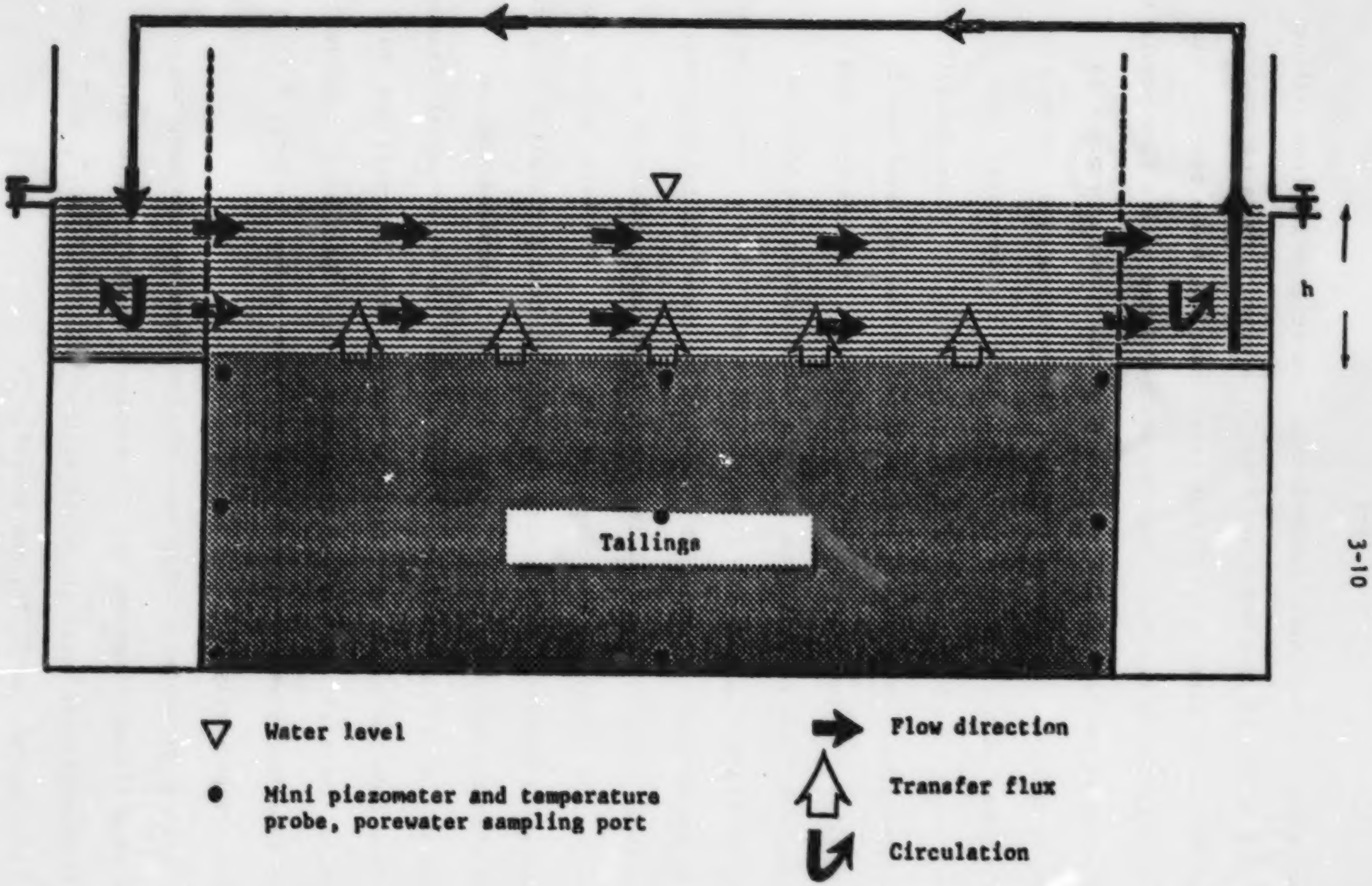
h
Tailings
▽ Water level
● Mini piezometer and temperature probe, porewater sampling port
Flow direction
Transfer flux
Circulation

Fig. 4 - Underwater buried tailings lysimeter - no flow configuration.

THEORY

CONTINUOUS FLOW - STEADY-STATE CONDITION

Assuming that there exists an equilibrium between tailings solid phase and porewater concentrations of a certain constituent such as Ra-226, Ca, total dissolved solids, etc., the mass transfer flux 'J' from tailings porewater to the over-laying waterbody above the interface, in the absence of any porewater flow, is given by:

$$J = K(C_o - C) \qquad \text{Eq 1}$$

where, J = flux, mass per unit area per unit time, for Ra, $Bq/m^2/hr$.

C_o = concentration in tailings porewater below the interface, for Ra Bq/m^3.

C = concentration in the water column above the interface, for Ra Bq/m^3.

K = transfer coefficient, m/hr.

For a flow volume of Q m^3/hr, the concentration in the water column above the interface is given by the Equation:

$$J = \frac{CQ}{A} \qquad \text{Eq 2}$$

where, A = surface area of the tailings/water interface, m^2.

Hence, by measuring the porewater concentration C_o and the concentration 'C' in the water column for a given flow Q in steady-state condition, both J and K can be measured experimentally.

NO-FLOW CONDITION

In the case of a stagnant water column of height 'h'm above the tailings surface with no flow, the mass transfer flux, J(t), is time-dependent given by:

$$J(t) = K[C_o - C(t)] \qquad \text{Eq 3}$$

where, $C(t)$ = time-dependent concentration in the water column above the interface, for Ra, Bq/m^3.

The rate of change of concentration $C(t)$ from mass conservation is:

$$\frac{dC(t)}{dt} = \frac{\text{Flux} \cdot \text{Area}}{\text{Volume of water column}} = \frac{\text{Flux}}{\text{Effective height of water column}}$$

i.e., $$\frac{dC(t)}{dt} = \frac{J(t)}{h} = \frac{K}{h}(C_o - C(t))$$

i.e., $$\frac{dC(t)}{dt} + \frac{K}{h}C(t) - \frac{K}{h}C_o = 0 \qquad \text{Eq 4}$$

which is a first order differential equation with solution of the form:

$$C(t) = A + B\ e^{-\alpha t} \qquad \text{Eq 5}$$

Applying the boundary conditions:

$$C(t) = 0 \quad \text{at } t = 0$$

$$\text{and} \quad C(t) \rightarrow C_o \text{ at } t \rightarrow \infty$$

the time-dependent solutions of $C(t)$ and $J(t)$ are:

$$C(t) = C_o[1 - \exp(-\frac{K}{h}t)] \qquad \text{Eq 6}$$

$$J(t) = J_o \exp(-\frac{K}{h}t) \qquad \text{Eq 7}$$

where, $J_o = K\ C_o$, flux at time $t = 0$ $\qquad$ Eq 8

From measurements of $C(t)$ as a function of time, K, C_o and J_o can be measured experimentally.

RESULTS

For the steady-state flow condition, the results obtained for the transfer flux 'J' and transfer coefficient 'K' are given, respectively, for Ra-226 and total dissolved solids in Table 1 and 2. Experiments were performed for a period of nine months for different flow volumes 'Q' and two water column heights 'h'. Figure 5 shows for Ra-226 the variation of mass transfer flux J, porewater concentration C_o and the transfer coefficient K as a function of time. The value of the transfer flux decreased by a factor of 12 from 4.48 $Bq/m^2/hr$

Table 1 - Continuous steady-state flow experiment, data for flow rate, flow velocity, Ra-226 concentrations in the water column above the tailings, tailings pore water concentrations and calculated values of the transfer flux J and transfer coefficient K for Ra-226.

No.	Time in Days From Start	Average Flow Rate Q mL/min	Flow Velocity v cm/min	Ra-226 Conc. in Water Above Tailings		Ave. Ra-226 Conc. in Tailings Porewater		Transfer Flux J		Transfer Coeff. K	
				$Bq/m^3 x10^3$	$pCi/m^3 x10^3$	$Bq/m^3 x10^3$	$pCi/m^3 x10^3$	$Bq/m^2/hr$	$pCi/m^2/hr$	$m/hr x10^{-4}$	m/month
1	20	46.65	0.085	0.440	12.0	9.86	266.2	4.51	121.8	4.8	0.35
2	27	66.20	0.121	0.370	10.0			5.34	144.1	5.6	0.40
3	35	20.65	0.038	0.480	13.0	Bottom Average		2.16	58.4	3.5	0.26
4	51	32.00	0.058	0.330	9.0	7.41	200.1	2.32	62.7	3.7	0.27
5	64	88.90	0.162	0.110	3.0	Middle Average		2.15	58.0	3.3	0.24
6	78	105.28	0.192	0.037	1.0	6.58	177.7	0.85	22.9	1.3	0.09
7	85	52.00	0.095	0.170	4.6			1.93	52.1	3.0	0.22
		Water column height for the set above was h=0.178 m				4.96	134 bottom	-	-	-	-
8	141	18.54	0.119	0.330	9.0	Bottom		1.34	36.3	5.3	0.38
9	146	3.36	0.021	0.810	22.0			0.60	16.1	2,9	0.21
10	149	13.65	0.087	-	-			-	-	-	-
11	238	7.93	0.051	0.333	9.0	Bottom Average		0.57	15.5	2.3	0.16
12	239	41.53	0.265	0.074	2.0	4.44	120.0	0.67	18.1	2.4	0.17
13	240	14.00	0.089	0.022	6.0			0.68	18.3	2.6	0.18
14	245	26.24	0.168	0.150	4.0			0.84	22.8	3.1	0.22
15	246	75.50	0.483	0.037	1.0	Middle Average		0.61	16.4	2.1	0.15
16	253	38.58	0.247	0.074	2.0	2.870	77.5	0.62	16.8	2.2	0.16
17	254	74.73	0.478	0.074	2.0			1.20	32.5	4.3	0.31
18	254	117.60	0.752	0.004	0.1			0.01	2.6	0.3	0 02
19	257	356.80	2.281	0.006	0.15			0.043	11.6	1.5	0.11
20	259	213.00	1.361	0.074	2.0			-	-	-	-
21	259	234.13	1.497	0.007	0.2			0.38	10.2	1.3	0.09
22	260	184.80	1.181	0.017	0.45			0.67	18.1	2.3	0.17
23	261	265.44	1.697	0.004	0.1			0.21	5.8	0.7	0.05
24	261	323.50	2.068	0.004	0.1			0.26	7.0	0.9	0.07
25	264	279.87	1.789	0.0	0.0			-	-	-	-
26	265	137.76	0.881	0.019	0.5			0.56	15.0	1.9	0.14
		Water column height for the above set was h=0.055 m				3.93	106 bottom				
						Bottom					

Table 2 - For continuous steady-state flow experiment, data for flow rate, flow velocity, TDS in water column above the tailings, tailings porewater concentrations and calculated values of transfer flux J and transfer coefficient K for total dissolved solids.

No.	Time in Days from Start	Average Flow Rate Q mL/min	Flow Velocity v, cm/min	TDS in Water Column Above Tailings $mg/m^3 \times 10^3$	Average TDS in Tailings Porewater $mg/m^3 \times 10^3$	Transfer Flux J $mg/m^2/hr$	Transfer Coefficient K $m/hr \times 10^{-4}$	Transfer Coefficient K m/month
1	20	46.65	0.085	38.95	2523.3 (bottom)	395.4	1.94	0.14
2	27	66.20	0.121	31.51		454.0	2.22	0.16
3	35	20.65	0.038	65.70	1639.9 (middle)	295.3	1.46	0.11
4	64	88.9	0.162	15.18		293.7	1.42	0.10
5	78	105.3	0.192	6.50	2081.0 (average)	148.9	0.72	0.05

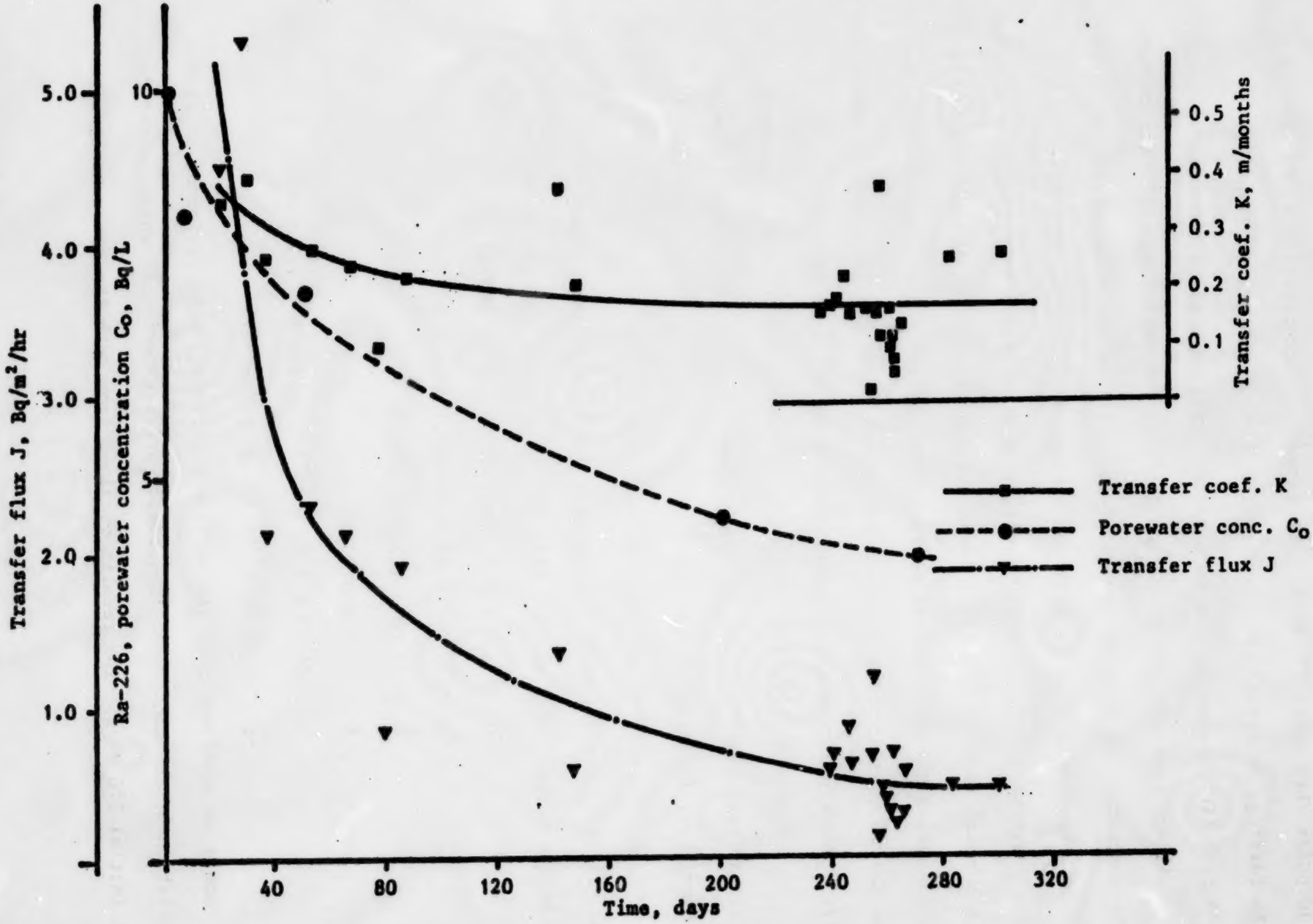

Fig. 5 - Variations of the transfer flux J, porewater concentration C_o and transfer coefficient K for Ra-226 as a function of time.

(121.8 pCi/m²/hr) at the start of the experiment to an average of 0.37 Bq/m²/hr (10 pCi/m /hr) after 270 days. The porewater concentration at the bottom of the tailings decreased by a factor of 2.5, from 9.9 Bq/L (267 pCi/L) to 3.9 Bq/L (105 pCi/L) for the same period. Figure 6 shows the time variation of tailings porewater concentration profiles for Ra-226 at different depths. The concentration profiles changed from a uniform distribution at the start of the experiment to a linearly increasing concentration with depth as the time progressed. The transfer coefficient K decreased by a factor of 2, from 0.4 m/month (5.62×10^{-4} m/hr) to an average value of 0.18 m/month (2.56×10^{-4} m/hr) during the experimental period. The correlation between the flow velocity and Ra-226 concentration in the water column above the interface is shown in Fig. 7 where a linear correlation breaks down for very high flows.

Figure 8 shows the time dependent variation of Ra-226 concentration C(t) and transfer flux J(t) for the stagnant water column for no-flow condition. A non-linear function of the form,

$$C(t) = C_{o1} - C_{o2} \exp(-\frac{K}{h}t) \qquad \text{Eq 9}$$

was fitted to the observed data using a non-linear least square fitting technique (Appendix A) and the best values obtained for the parameters were:

$C_{o1} = 1.39 \pm 0.57$ Bq/L (37.6 ± 15.3 pCi/L)

$C_{o2} = 1.36 \pm 0.54$ Bq/L (36.7 ± 14.5 pCi/L)

$\frac{K}{h} = (2.00 \pm 1.26) \times 10^{-3}\ \text{hr}^{-1}$

$K = 0.34 \pm 0.21$ m/month

$(4.73 \pm 3.0) \times 10^{-4}$ m/hr for $h = 0.236$ m (0.178×1.33)

$J_o = 0.65$ Bq/m²/hr (17.44 pCi/m²/hr)

It should be mentioned that the no-flow experiments were performed after the completion of the continuous flow steady-state experiments which lasted for the initial 270 days. The results obtained from both the experiments were

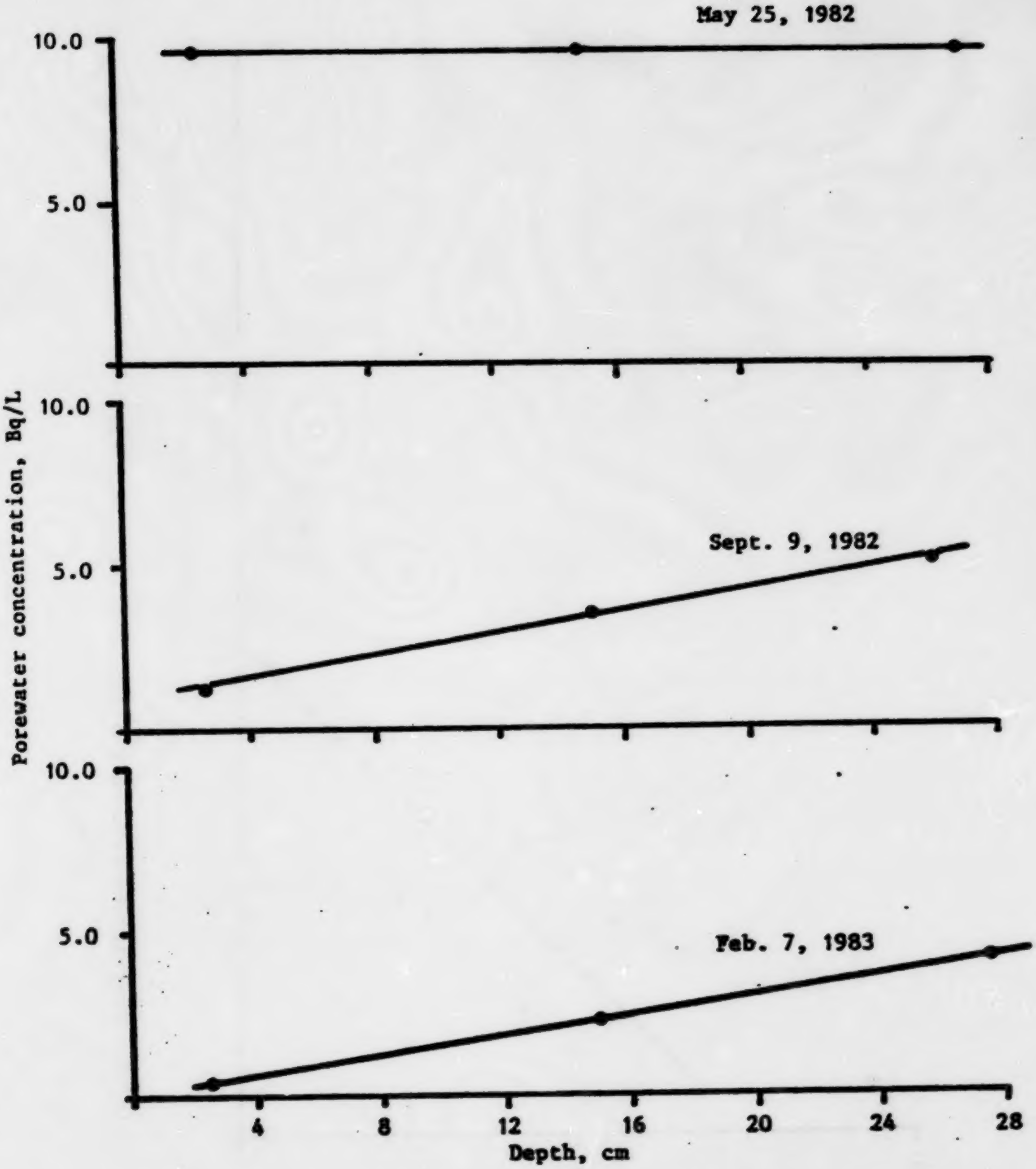

Fig. 6 – Time variation of tailings porewater concentration profile for Ra-226 at various depths.

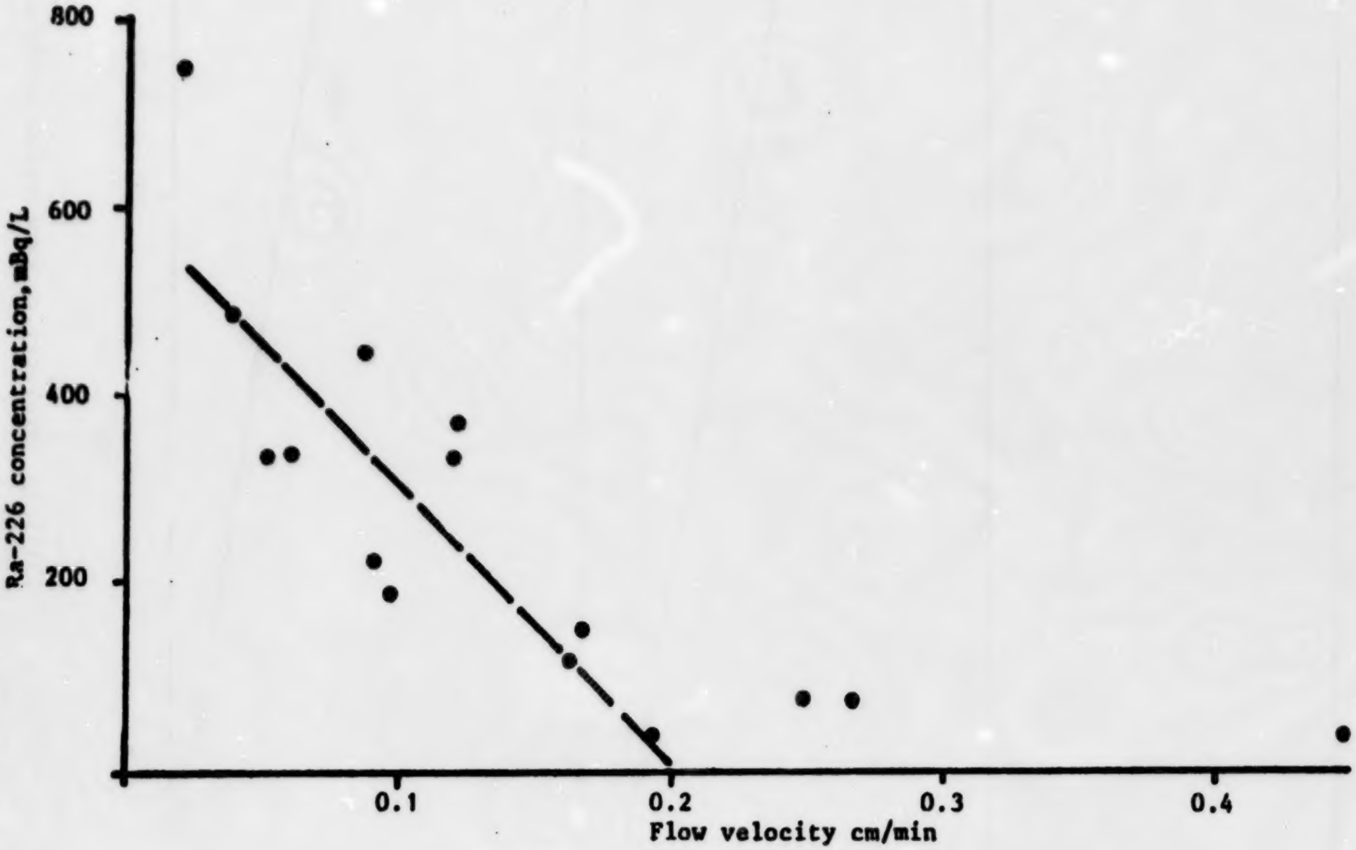

Fig. 7 - Correlation between the flow velocity and Ra-226 concentration in the water column.

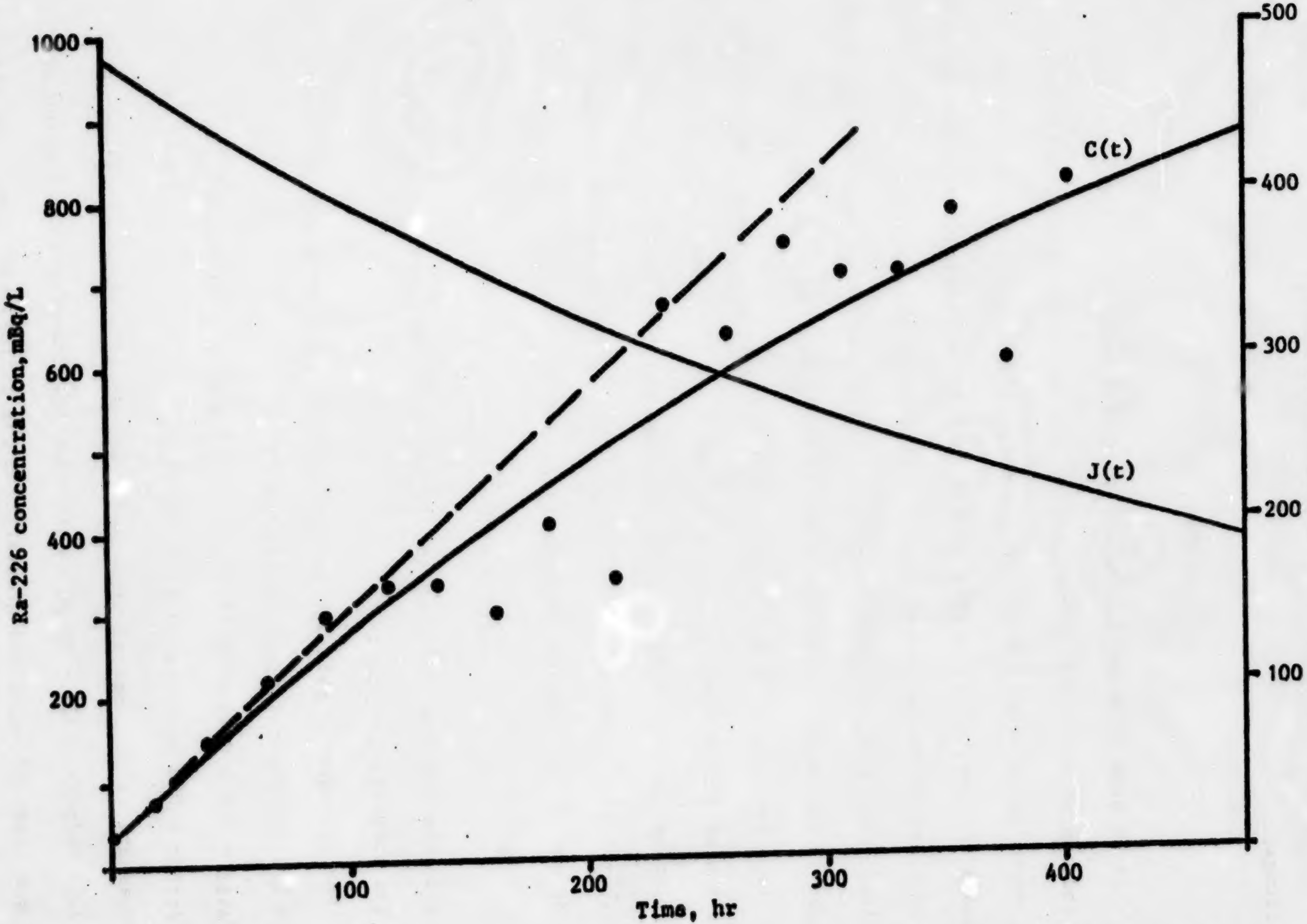

Fig. 8 - Variation of Ra-226 concentration C(t) and transfer flux J(t) as a function of time in the stagnant water column above tailings. Dotted line represents initial slope of C(t).

comparable. In all the experiments no temperature or piezometric pressure gradients were observed signifying no porewater transportation in or out of the tailings.

DISCUSSION

It is seen from the laboratory lysimeter results that for tailings buried underwater the transfer coefficient and flux of Ra-226 from tailings to the over-laying water strata and the porewater concentration of Ra-226 all decreased with time. The transfer flux J showed the greatest reduction because of the decrease in the porewater concentration below the interface from initial high values. The tailings porewater concentration no longer remained constant as assumed in the model (Equations 1 and 3) but exhibited a concentration profile characteristic of a diffusion controlled process. Using the measured value of the diffusion coefficient, $D_o = 8.89 \times 10^{-10}$ m²/sec for Ra-226 in dilute solutions at 25°C (18), and the observed tailings porewater concentration gradient, $\frac{dc}{dz} = 15.7 \times 10^3$ Bq/m⁴ (424×10^3 pCi/m⁴), the molecular diffusion flux J_{diff} out of tailings can be calculated as:

$$J_{diff} = -\phi\, D \left.\frac{dc}{dz}\right|_{z=0} \qquad \text{Eq 10}$$

where, ϕ is the tailings porosity and J_{diff} is in the direction opposite to the increasing depth z.

For uncompacted tailings in the lysimeter, the porosity is expected to be $\phi \simeq 0.6$ and the calculated diffusion flux:

$$J_{diff} = 0.6 \times 8.89 \times 10^{-10} \times 15.7 \times 10^3 = 8.37 \times 10^{-6} \text{ Bq/m}^2\text{/sec}$$

i.e., $J_{diff} = 0.030$ Bq/m²/hr (0.81 pCi/m²/hr).

These values are to be compared with the experimentally observed values of 4.48 to 0.37 Bq/m²/hr (121 - 10 pCi/m²/hr) for the transfer flux J which are at least one order of magnitude greater than the diffusional flux. The results demonstrate that though there is a small diffusional component to the total flux, a greater component of the flux can be attributed to mass dissolution from the solid phase to the immediate layer in contact and its transfer to the water column in contact either directly or via porewater. The observed decrease in the transfer flux, porewater concentration and transfer coefficient with time is probably because of the shifting equilibrium between solid and liquid phases.

The measured values of the mass transfer coefficient $K = 0.4$ to 0.18 m/month are comparable to those calculated by Halbert et al. (17) for uranium tailings buried underwater in a lake scenario. It should be mentioned that the mass transfer coefficient was calculated from Equations 1 or 3 using the observed values of J, C_0 and C, and where, for porewater concentration, a time average value of C_0 within the tailings was used.

It is also seen from Fig. 7 that there exists a correlation between the flow velocity in the water column and its Ra-226 concentration for moderate flow rates. At higher flow rates the relationship breaks down which is partly attributed to experimental errors involved in detecting low Ra-226 concentrations ~3.7 mBq/L (0.1 pCi/L).

The values obtained for the transfer coefficient for total dissolved solids (TDS) $K = 0.16$ to 0.05 m/month are about one third of those obtained for Ra-226, suggesting that these two components of tailings solid and liquid phase migrate differently.

ACKNOWLEDGEMENT

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Appendix A - Non-linear least square fit program

```
C     MAIN PROGRAM NON LINEAR CURVEFIT
      PROGRAM NCURVE(OUTPUT,TAPE5,TAPE6=OUTPUT)
      DIMENSION W(320),X(3),E(6),F(100),FJ(100),Y(100),T(100),TITLE(20)
      COMMON  /MYCOM/Y,T
C     CALFUN
      N=3
      IOUT=6
      IINP=5
      REWIND IINP
      READ(IINP,200)(TITLE(I),I=1,13)
      WRITE(IOUT,190)(TITLE(I),I=1,13)
      I=0
  110 I=I+1
      READ(IINP,*) Y(I),T(I)
      WRITE(IOUT,170) Y(I),T(I)
      IF(Y(I))120,110,110
  120 M=I-1
      ESC=1.0E2
      IPRINT=1
      MAXIT=20
      READ(IINP,*) (X(I),E(I),I=1,N)
      WRITE(IOUT,170) (X(I),E(I),I=1,N)
      READ(IINP,*) HT
      WRITE(IOUT,180) HT
      CALL DFS001(M,N,F,X,E,ESC,IPRINT,MAXIT,W,IOUT)
      J=1
      DO 140 I=1,N
      E(I)=SQRT(W(J))
      J=J+N+1
  140 CONTINUE
      DO 150 I=1,M
      F(I)=Y(I)-F(I)
      FJ(I)=1000.*HT*X(2)*X(3)*EXP(-X(3)*T(I))
  150 CONTINUE
      WRITE(IOUT,160) (TITLE(I),I=1,13),(X(I),E(I),I=1,N),(T(I),Y(I),F(
     1I),FJ(I),I=1,M)
  160 FORMAT(*1*,40X,13A6,/,40X,*PARAMETERS*,20X,*STANDARD DEVIATION*,/
     1,3(1PE50.4,E35.4,/),23X,*T(I)*,20X,*Y(I)*,20X,*YCAL(I)*,
     220X,*J(I)*,/,(E31.4,3E25.4))
  170 FORMAT(2E15.5)
  180 FORMAT(E15.5)
  190 FORMAT(*1*,40X,13A6)
  200 FORMAT(13A6)
      END
      SUBROUTINE CALFUN(M,N,F,X)
      DIMENSION X(3),F(100),Y(100),T(100)
      COMMON /MYCOM/Y,T
      DO 1 I=1,M
      F(I)=Y(I)-X(1)+X(2)*EXP(-X(3)*T(I))
    1 CONTINUE
      RETURN
      END
```

NONLINEAR LEAST SQUAREFIT FOR RA-226 DATA, NO FLOW CONDITION

```
 .10000E+01   0.
 .20000E+01    .17000E+02
 .40000E+01    .41000E+02
 .60000E+01    .65000E+02
 .60000E+01    .91000E+02
 .90000E+01    .11500E+03
 .90000E+01    .13700E+03
 .80000E+01    .16100E+03
 .11000E+02    .18500E+03
 .90000E+01    .20900E+03
 .18000E+02    .23300E+03
 .17000E+02    .25900E+03
 .20000E+02    .28350E+03
 .19000E+02    .30600E+03
 .19000E+02    .33100E+03
 .21000E+02    .35400E+03
 .16000E+02    .37800E+03
 .22000E+02    .40200E+03
 .23000E+02    .47400E+03
-.10000E+01   -.10000E+01
 .45000E+02    .10000E-02
 .45000E+02    .10000E-02
 .20000E-02    .10000E-03
 .23700E+00
```

```
ITERATION    0          4 CALLS OF CALFUN       F=        1.81477130E+02
PARAMETERS
 4.50000000E+01         4.50000000E+01          2.00000000E-03
FUNCTIONS
 1.00000000E+00         4.95717709E-01          4.57238141E-01         5.14294391E-01         5.12060319E-01
-2.45967887E-01        -1.78505661E+00         -4.38858144E+00        -2.91695512E+00        -6.37369972E+00
 1.23782778E+00        -1.19301495E+00          5.25105309E-01        -1.59806360E+00        -2.78816008E+00
-1.83171886E+00        -7.87066224E+00         -2.86091429E+00        -4.56181242E+00

ITERATION    1          6 CALLS OF CALFUN       F=        8.01217076E+01
PARAMETERS
 3.75914767E+01         3.67131602E+01          2.00395280E-03
FUNCTIONS
 1.21683484E-01        -1.07966708E-01          2.25847250E-01         6.37899689E-01         1.00165636E+00
 5.65105939E-01        -6.92395812E-01         -3.00244262E+00        -1.25097405E+00        -4.44087865E+00
 3.42509065E+00         1.25656960E+00          3.20980614E+00         1.29272878E+00         3.21095600E-01
 1.46918141E+00        -4.37888623E+00          8.12868537E-01        -3.91197141E-01

ITERATION    2         10 CALLS OF CALFUN       F=        8.01214112E+01
PARAMETERS
 3.76439781E+01         3.67628633E+01          2.00043453E-03
FUNCTIONS
 1.18885199E-01        -1.10304494E-01          2.24013593E-01         6.36412439E-01         1.00038226E+00
 5.63892540E-01        -6.93657898E-01         -3.00386138E+00        -1.25264756E+00        -4.44289545E+00
 3.42265090E+00         1.25359128E+00          3.20625228E+00         1.28859476E+00         3.16265739E-01
 1.46366941E+00        -4.38514744E+00          8.05824976E-01        -4.00740778E-01

ITERATION    3         13 CALLS OF CALFUN       F=        8.01213660E+01
PARAMETERS
 3.76089416E+01         3.67315204E+01          2.00359240E-03
FUNCTIONS
 1.22578782E-01        -1.07468846E-01          2.25794298E-01         6.37308047E-01         1.00049491E+00
 5.63431260E-01        -6.94530184E-01         -3.00506675E+00        -1.25407575E+00        -4.44444578E+00
 3.42107013E+00         1.25207137E+00          3.20487045E+00         1.28740204E+00         3.15346241E-01
 1.46305413E+00        -4.38539691E+00          8.05985462E-01        -3.99133255E-01
```

```
ITERATION   4        15 CALLS OF CALFUN     F=      8.01213487E+01
PARAMETERS
 3.75964288E+01       3.67180089E+01       2.00447181E-03
FUNCTIONS
   .580 CP SECONDS COMPILATION TIME
 1.21580123E-01      -1.08545588E-01       2.24641732E-01       6.36116744E-01       9.99299654E-01
 5.62264150E-01      -6.95647197E-01      -3.00610509E+00      -1.25501273E+00      -4.44526106E+00
 3.42039479E+00       1.25156591E+00       3.20454042E+00       1.28724468E+00       3.15392113E-01
 1.46329619E+00      -4.38494202E+00       8.06660241E-01      -3.97767251E-01

ITERATION   5        17 CALLS OF CALFUN     F=      8.01213486E+01
PARAMETERS
 3.75963925E+01       3.67179712E+01       2.00447281E-03
FUNCTIONS
 1.21578681E-01      -1.08546369E-01       2.24641879E-01       6.36117814E-01       9.99301716E-01
 5.62267118E-01      -6.95643406E-01      -3.00610041E+00      -1.25500718E+00      -4.44525465E+00
 3.42040206E+00       1.25157408E+00       3.20454942E+00       1.28725444E+00       3.15402695E-01
 1.46330751E+00      -4.38492994E+00       8.06673063E-01      -3.97752303E-01

DFSOO1 FINAL VALUES OF FUNCTIONS AND VARIABLES

ITERATION   5        17 CALLS OF CALFUN     F=      8.01213486E+01
PARAMETERS
 3.75963925E+01       3.67179712E+01       2.00447281E-03
FUNCTIONS
 1.21578681E-01      -1.08546369E-01       2.24641879E-01       6.36117814E-01       9.99301716E-01
 5.62267118E-01      -6.95643406E-01      -3.00610041E+00      -1.25500718E+00      -4.44525465E+00
 3.42040206E+00       1.25157408E+00       3.20454942E+00       1.28725444E+00       3.15402695E-01
 1.46330751E+00      -4.38492994E+00       8.06673063E-01      -3.97752303E-01
```

```
          NONLINEAR LEAST SQUAREFIT FOR RA-226 DATA, NO FLOW CONDITION
        PARAMETERS                          STANDARD DEVIATION
        3.7596E+01                             1.5294E+01
        3.6718E+01                             1.4520E+01
        2.0045E-03                             1.2561E-03
  T(I)              Y(I)                YCAL(I)               J(I)
0.                  1.0000E+00          8.7842E-01          1.7443E+01
1.7000E+01          2.0000E+00          2.1085E+00          1.6859E+01
4.1000E+01          4.0000E+00          3.7754E+00          1.6067E+01
6.5000E+01          6.0000E+00          5.3639E+00          1.5312E+01
9.1000E+01          8.0000E+00          7.0007E+00          1.4535E+01
1.1500E+02          9.0000E+00          8.4377E+00          1.3852E+01
1.3700E+02          9.0000E+00          9.6956E+00          1.3255E+01
1.6100E+02          8.0000E+00          1.1006E+01          1.2632E+01
1.8500E+02          1.1000E+01          1.2255E+01          1.2039E+01
2.0900E+02          9.0000E+00          1.3445E+01          1.1473E+01
2.3300E+02          1.8000E+01          1.4580E+01          1.0934E+01
2.5900E+02          1.7000E+01          1.5748E+01          1.0379E+01
2.8350E+02          2.0000E+01          1.6795E+01          9.8817E+00
3.0600E+02          1.9000E+01          1.7713E+01          9.4459E+00
3.3100E+02          1.9000E+01          1.8685E+01          8.9842E+00
3.5400E+02          2.1000E+01          1.9537E+01          8.5794E+00
3.7800E+02          1.6000E+01          2.0385E+01          8.1765E+00
4.0200E+02          2.2000E+01          2.1193E+01          7.7924E+00
4.7400E+02          2.3000E+01          2.3398E+01          6.7452E+00
```

Discussion of Paper by N. Dave

A. Oliver	Can you comment on the effect of water cover and oxidation due to pyrite?
N. Dave	There is no evidence of oxidation to date, but some decrease of pH from 6.0 to 5.3. The pH of the water is 5.5.
D. Francis	Why use distilled water instead of lake water?
N. Dave	We thought it was better to use tap water or lake water, but there was too much dissolved air, and the air bubbles were forming in the tank.
L. Heymann	Did you measure the oxygen in the water as a function of depth?
N. Dave	No
E. Barnes	When the lysimeters were filled, were the solids decanted first?
N. Dave	Both the settled and decanted solids were used.
E. Barnes	You reported 270 pCi/l of pore water. This number is rather surprising as we normally expect 3000 pCi/l Ra^{226}.
N. Dave	We were also surprised by this figure. Some of the radium could possibly be adsorbed.
E. Barnes	If we are going to deep lake disposal, people will assume low fluxes.
N. Dave	We have measured the transfer coefficient between the pore water and the upper and lower zones. This is based on the assumption that the tailings have settled. The transfer coefficient of 0.3-0.4 is very similar to that calculated by Senes as 0.5 m^2/month.
T. Goode	It appears that the rate of transfer will depend on how the tailings were deposited. There will be areas of fines and areas of coarse particles.
N. Dave	Yes, I agree that the coefficient will vary. If there is sufficient in the deep lake disposal method, then actual field conditions should be tested.

Discussion of Paper by N. Dave cont'd

V. Haw	Is the transfer coefficient independent of pore water? Was the Ra^{226} content of the solids measured?
N. Dave	The transfer coefficient is based on an equilibrium between the solid and liquid phases. Yes, the solids were measured for Ra^{226}, and they varied over the range 280-320 pCi/g Ra^{226}.
E. Barnes	Regarding Nordic tailings, what was the plume?
N. Dave	At low pH, of <4, the Ra^{226} was 50-100 pCi/l. In the control pH 4.5 Ra^{226} was 5-10 pCi/l. In the neutral zone, it was <5 pCi/l.
E. Barnes	Would you agree that Nordic is not typical of other tailings?
N. Dave	Yes, it is the worse site. There is maximum oxidation, the dam is pervious, and the water is moving in the ground water aquifer.
B. Chomyn	What is the size or extent of the contaminant plume?
N. Dave	The extent is 15 m from the dam, which is 1 meter/year movement. This compares to Ca, SO_4 of 300 meters.

Report of
CUPMC Analytical Subcommittee

by

Lloyd Dalton
CANMET
Chairman, CUPMC Analytical Subcommittee, 1982-83

Presented at
Canadian Uranium Producers
Metallurgical Meeting
Mississauga, Ont.
May 19, 1983

CUPMC ANALYTICAL SUBCOMMITTEE

Just to jog your memory, the analytical subcommittee was reactivated in 1977 by the metallurgical committee because of the pressure of radiochemical work. Since then the subcommittee has been meeting every 6 months and working on various projects. The last report to the metallurgical committee was presented in the spring of 1980 by Ron Blockmore.

The subcommittee has carried out three round-robin analytical exercises since 1980. The most recent was that of fluoride determination in yellowcake. This arose because of a disagreement between laboratories, which was resolved by the round-robin. A spinoff from this exercise was methodology information on the use of the fluoride specific ion electrode, specifically the type of buffer solution.

Another round-robin project was conducted by the CCIW for Th-230 and Pb-210 determination on samples from the Elliot Lake Pilot Plant Project.

The round-robin that we were concerned with in 1980-81 was that of Ra-226 determination in CANMET reference materials Dl-1 and Dl-1A. There were two goals to this exercise, firstly to have a material that we would know the Ra-226 content with some certainty and secondly to demonstrate our competency. Unfortunately, the design of the round-robin was not quite right and to some extent we failed to reach our objectives. However it should be stated that the 95% confidence limits of the analytical data was within the calculated Ra-226 content, based on the uranium determination for Dl-1A and very close in Dl-1, as is shown in the first slide.

At the last report to the Metallurgical Committee, it was mentioned that several organizations in the analytical subcommittee were participating in the U.S. Environmental Protection Agency Ra-226 laboratory monitoring program. Unfortunately, this practice has virtually ceased because of problems experienced in shipping radioactive materials. To my knowledge there is no Canadian counterpart to this EPA activity.

Earlier in my report I mentioned that our round-robin for Ra-226 did not quite achieve the goals which we had set. This left us with a problem. Do we go through another expensive round-robin or seek alternative action. The ideal arrangement would be to obtain material which is known to be in secular equilibrium, so that the activities of all the isotypes in the uranium decay series could be calculated from the uranium content.

An american radiochemist, Claude Sill, has published data which indicated that a pitchblende ore that he had obtained from the Schwartzwalder deposit in Grand Junction, Colorado, was in secular equilibrium. Other workers throughout the world have published data to support Sill's contentions. Because this ore has had a good deal of radiochemistry done on it, it appeared to the members of the subcommittee that we should obtain this ore and use it as a reference material. I do not want to convey the impression that there aren't any Canadian ores which would serve our purposes but the documentation supporting the radiochemical virtues of this ore warranted on effort to have this ore as a reference material.

Accordingly, a geologist was engaged, sent to Colorado and supervised the selection of four different lots of ore from the Schwartzwalder deposit. He returned to Ottawa with subsamples of these four samples for preliminary radiochemical work before we made our final selection. A problem developed insofar as we were not able to obtain our samples from exactly the same spot as Sill had obtained his ore, but the preliminary analytical work indicates that the ore will be suitable for our purposes.

We expect about 500 lbs. of ore to be shipped to CANMET this summer, where it will be processed, bottled, and stored. For monetary purposes, it will be another CANMET reference material. The subcommittee will carry out a round-robin uranium determination and then we will be in a position to calculate the activities of all the isotopes in the uranium decay series and have a reference material which can be used for calibration and evaluation purposes. This activity has been of the major efforts of the subcommittee.

It should be mentioned that there is a surge of activity in radiochemical reference materials at the present time. CANMET is pursuing the certification of 4 of its uranium reference materials for Ra-226, Th-230 and Pb-210 content and is also establishing analyzed materials for the National Uranium Tailings Project. Also, there are several powder materials available from the U.S. E.P.A., but only in very small quantities. The availability of radiochemical reference materials looks very much brighter than it did a year ago.

The last, and certainly not the least topic which I want to bring up is that of the reporting of analytical values. C. Miedma of Eldorado R&D, who was chairman of the subcommittee in 1981-82, prepared a document on suggested reporting practice of analytical values. This document produced a great deal of discussion and, as of the present time, the subcommittee has not taken any firm action insofar as the subcommittee as a whole is concerned. That is all members of the subcommittee are not following the recommendations of the document.

I think that it is workwhile to go over some of the document because I feel that the reporting style which the document advocates will eventually be put into general use.

A major recommendation of the report is that an uncertainty is reported along with an analyical value, as the determined analytical value is not an invariant number.

The following style is suggested as is shown on the next slide.

A ± B(C)

where A is the analytical result

B is the total random uncertainty

C is the total random and systematic uncertainty

The random uncertainty has a number of components, each of which makes a contribution to the total random uncertainty. But one of the most significant aspects of random uncertainty for radiochemistry is that of counting statistics. Quite simply, this uncertainty decreases as the number of total counts increases for a particular measurement and the uncertainty is equal to the square root of the number of counts.

In order to arrive at the total random uncertainty, one can make replicate measurements of calibration standards and blanks, but the total random uncertainty should be incorporated into a single value.

The manner in which all these uncertainties can be combined is by the application of the Law of Propogation of Error, which is as follows:

for a function $z = f(x,y)$

the error in z is given by

$$\sigma_z^2 = \left[\left(\frac{dx}{dx}\right)_y^2 \sigma_x^2 + \left(\frac{dz}{dy}\right)_x^2 \sigma_y^2 \right]$$

where the d's are the partial derivatives of z with respect to x and y, and the sigmas are the estimated variance terms for x and y.

At first glance, it would appear that one would have to do some mathematics in order to arrive at the uncertaintly in z, but there is a way around this, which is simply to divide the above equation by the function itself, with the result

$$\frac{\sigma_z}{z} = \left[\left(\frac{\sigma_x}{x}\right)^2 + \left(\frac{\sigma_y}{y}\right)^2\right]^{\frac{1}{2}}$$

There are a number of forms of the above equation depending upon the complexity of the function of z.

To my knowledge, very few results are reported in the industry in this fashion, especially results which are strictly for in-house use, the reason being is that of cost. It would take a certain amount of time to measure and calculate all the uncertainties which go into a determination and time generally being a precious commodity in an analytical lab, this sort of study just doesn't get done, however lofty the ideal. As mentioned previously, there is not uniform action by the subcommittee for this style of reporting analytical data, yet.

Another point raised in the document is that of reporting analytical values which are very close to the detection limit. A common way of handling this situation is to report "less than" a certain number. It is suggested that an analytical value be reported as calculated, even though it may be a negative result. This can easily arise when the background estimation is larger than the counts arising from the sample. There is even a theoretical reason for this, because there is a distribution associated with both these measurements and it is merely a case of overlapping distributions, as shown in the next slide.

Thus, there is a sound case for reporting analytical values as a negative result, but an uncertainty should be reported along with the negative value for proper interpretation.

A case where the reporting of negative values can be beneficial is that of monitoring a particular location on a continuing basis. Consider the following hypothetical data, which was taken from an article on water quality control.

Column 1 gives results in a heavily censored form, and the only conclusion that can be reached is that one doesn't know whether the constituent is present or not.

Column 2 has negative results censored, one could conclude that the mean concentration was 1.2 µg, with a standard error of the mean of 0.47 and 95% confidence limits for the mean of 0.14 µg and 2.26 µg. Since the confidence limits do not include zero, it would appear that the evidence supports the presence of the constituent.

Column 3 gives the results as obtained, this data gives a mean concentration of 0.5 µg a standard error of the mean of 0.719 and 95% confidence limits for the mean of -1.13 and 2.13 µg. Here the evidence is insufficient to support the presence of the constituent.

Thus, it is fairly obvious from this data, although hypothetical, that there are definite advantages to reporting of negative values. However, I understand that there may be a problem with the formats of some computer programs which will not accept negative numbers.

The preceding summarizes the activities of the analytical subcommittee for the past few years. In the future, I expect that we will be busy with the ore from the Schwartzwalder deposit, reporting practice procedures and laboratory statistical quality control procedures. No doubt, there will also be on-going round-robin determinations for problems that occur from time to time.

TABLE 1

ANALYTICAL SUBCOMMITTEE

RA-226 ROUND-ROBIN

MATERIAL	THEORETICAL		DETERMINED	
	MEAN	95% CONFIDENCE LIMITS	MEAN	95% CONFIDENCE LIMITS
DL-1	13.6	12.9 - 14.2	12.1	11.2 - 13.0
DL-1A	38.4	37.4 - 39.4	37.9	35.1 - 40.8

all results in pCi/g

theoretical values based on the uranium content and the assumption of secular equilibrium.

TABLE 2

SUGGESTED REPORTING PRACTICE

A ± B(C)

where A is the analytical result
B is the total random uncertainty
C is the total random and systematic uncertainty

TABLE 3

LAW OF PROPAGATION OF ERROR

$$Z = f(x,y)$$

$$\sigma_Z^2 = \left[\left(\frac{dz}{dx}\right)_Y^2 \sigma_X^2 + \left(\frac{dz}{dy}\right)_X^2 \sigma_Y^2 \right]$$

$$\frac{\sigma_Z}{Z} = \left[\left(\frac{\sigma_X}{X}\right)^2 + \left(\frac{\sigma_Y}{Y}\right)^2 \right]^{1/2}$$

TABLE 2

SUGGESTED REPORTING PRACTICE

$$A \pm B(C)$$

where A is the analytical result

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$$\frac{\sigma_Z}{Z} = \left[\left(\frac{\sigma_X}{X}\right)^2 + \left(\frac{\sigma_Y}{Y}\right)^2 \right]^{1/2}$$

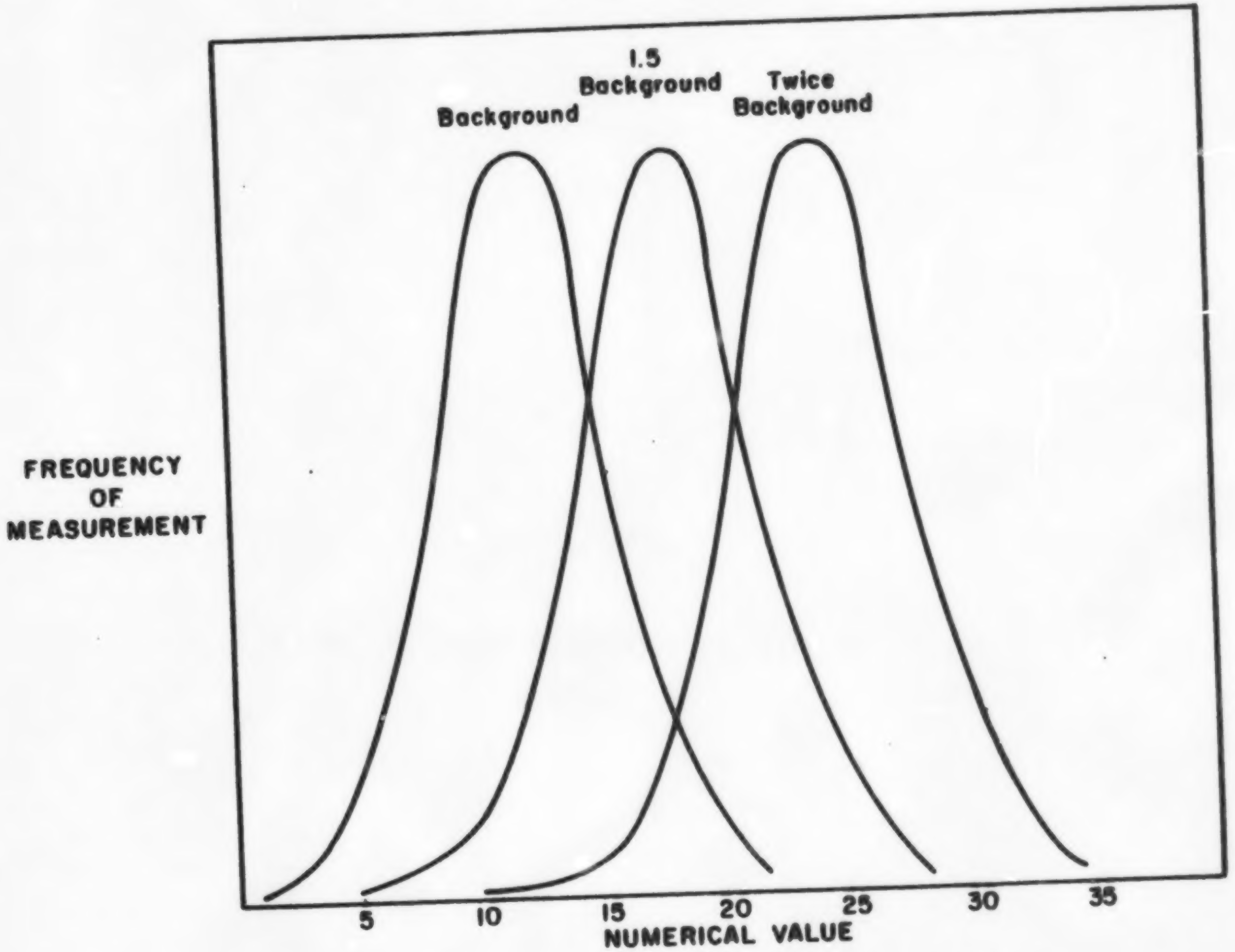

FIGURE 1: HYPOTHETICAL COUNT DISTRIBUTION

TABLE 4

HYPOTHETICAL ANALYTICAL DATA

	<3	2	2
	<3	0	-2
	<3	0	-1
	4	4	4
	3	3	3
	<3	0	-3
	<3	1	1
	<3	0	-1
	<3	0	0
	<3	2	2
mean	--	1.2	0.5
95% confidence limits	--	0.14 and 2.26	-1.13 and 2.13

all data in micrograms/liter

Discussion of Paper by L. Dalton

Question:	A comment - there should be uniformity on Becquerels and pCi/g. The result from a round robin was the question of sample preparation and how the determination was performed.
L. Dalton	In the 1981 methods, dissolution procedures were better. Each result should have the same weighting.
D. Francis	Is it possible that bias is part of the problem? Reporting less than (agree) should be eliminated and it is a good step in the reporting practice.

URANIUM WASTE MANAGEMENT
IN CANADA

by

L.A. Melis, P.Eng.

May 1983

for presentation at the Canadian Uranium Producers Metallurgical Committee Meeting in Toronto, Ontario, May 19, 20, 1983.

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URANIUM WASTE MANAGEMENT IN CANADA

ABSTRACT

A National Uranium Tailings Program has been established in Canada to develop research on the long-term abandonment of uranium mine tailings. A report was prepared and submitted which provided recommendations for potential research avenues for the disposal technology component of the Program. This paper describes certain sections of this report including a description of uranium tailings in Canada subdivided into inactive sites, active sites, new tailings sites and areas of tailings in a close-out situation. Effluent treatment practices and potential impoundment techniques for Canadian operations are also discussed.

INTRODUCTION

Uranium tailings have been the subject of much attention over the last decade due to their inherent environmental problems. The main items of concern have been the radionuclide content of tailings and its potential release to the environment as well as the acid-generating capacity of tailings for certain tailings impoundments.

A National Technical Planning Group on Uranium Tailings Research was organized by CANMET in 1980 to study the problem on a national basis. From their recommendations in a report issued in September 1981, the National Uranium Tailings Program (N.U.T.P.) was established along the guidelines recommended by the Planning Group for developing research on the long-term abandonment of uranium mine tailings.

Recommendations in the report concluded that the N.U.T.P. should consist of three main components; a sampling program, a modelling program and a disposal technology program.

A report has been prepared and submitted to the N.U.T.P. which describes uranium tailings management in Canada and details waste disposal technologies pertinent to tailings management. From this information recommendations were made with respect to potential research avenues for the disposal technology component of the N.U.T.P.

As a result of this work information pertinent to uranium tailings management in Canada has been assimilated. This report provides a description of each tailings site in Canada as well as effluent treatment practices for various operators and summarizes impoundment techniques that are being used or proposed for future operations. Research recommendations to the N.U.T.P. are not within the scope of this paper and are therefore not discussed.

There are several types of uranium tailings areas in Canada which require consideration. These include areas of inactive tailings, areas of tailings in a close-out situation, active tailings sites and new tailings sites for new operations coming on stream. The different tailings sites in Canada are described below. The location of uranium tailings areas and some of the significant uranium deposits in Canada are shown in Figure 1.

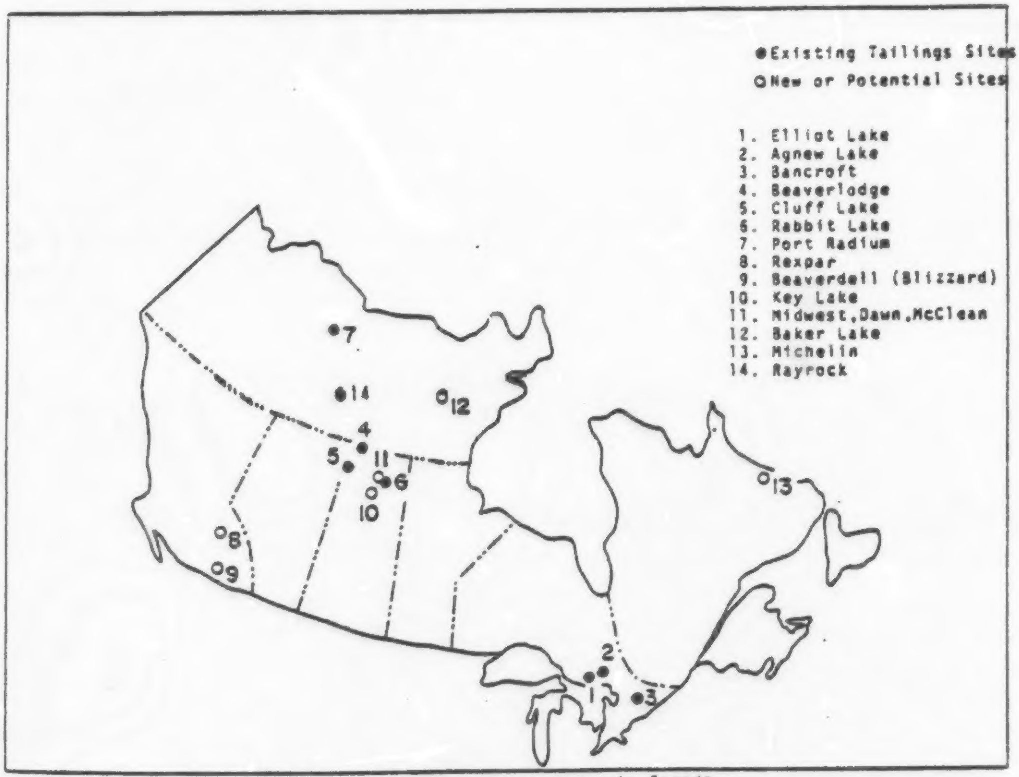
● Existing Tailings Sites
○ New or Potential Sites
1. Elliot Lake
2. Agnew Lake
3. Bancroft
4. Beaverlodge
5. Cluff Lake
6. Rabbit Lake
7. Port Radium
8. Rexpar
9. Beaverdell (Blizzard)
10. Key Lake
11. Midwest, Dawn, McClean
12. Baker Lake
13. Michelin
14. Rayrock
7
14
12
4
5
11
6
10
8
9
2
1
3
13

Figure 1 - Location of Uranium Areas in Canada
(adapted from Planning Group Report, September 1981)

INACTIVE TAILINGS

Inactive tailings in Canada include the Lorado and Gunnar tailings near Beaverlodge in northern Saskatchewan, the Port Radium and Rayrock tailings in the North West Territories, the Bicroft and Dyno areas near Bancroft, Ontario and the Nordic, Lacnor, Stanrock, Spanish American and Pronto tailings sites in the Elliot Lake area of northern Ontario. A list of the inactive tailings sites is provided in Table 1.

TABLE 1

INACTIVE TAILINGS SITES

Site	Date of Operation	Total Tonnes (Approx.)	Approx. Area (Hectares)
Lorado, Sask.	1957-1960	360,000	14
Gunnar, Sask.	1955-1964	5,500,000	45
Port Radium, N.W.T.	1953-1960	910,000	?
Rayrock, N.W.T.	1957-1959	80,000	?
Bicroft, Ont.	1958-1963	2,000,000	8
Dyno, Ont.	1958-1960	360,000	3
Faraday, Ont.	1957-1962	1,800,000	16
Nordic, Ont.	1957-1968	11,000,000	101
Lacnor, Ont.	1957-1960	2,700,000	24
Stanrock, Ont.	1958-1964	8,000,000	71
Spanish American, Ont.	1958-1959	450,000	5
Pronto, Ont.	1955-1966	2,100,000	47

Lorado

The Lorado tailings are located approximately 10 km southwest of Uranium City, Sask. on the west shore of Nero Lake. A custom uranium mill was operated from 1957 to 1960 by Lorado Uranium Mines Ltd. using an acid-leach circuit, ion exchange with NaCl elution and magnesium diuranate precipitation. The tailings were deposited at pH 2 directly to Nero Lake. Studies in this area have indicated the following site conditions at Lorado:

- the tailings are pyritic and acid generating (pH 3.5).
- there is extensive layering of coarse and fines due to the method of tailings deposition that was practiced.
- the exposed surface layers are red in colour and cemented with oxidized iron.
- natural revegetation is sparse.

- wind erosion is apparent on the drier, coarse areas.
- acid generation in submerged tailings is minimal.
- water erosion is apparent in the lower part of the tailings area.
- a precipitate plume high in radionuclides extends into Hansen Bay through a berm between Nero Lake and Beaverlodge Lake.
- the moss carpet on the lake bottom is acid tolerant and is still growing. It is acting as a concentration sink for some of the cations being released by the tailings.
- the tailings, moss carpet and Nero Lake water have reached a steady-state and are in equilibrium.
- Nero Lake has not undergone any major changes in the last 8 years.
- the berm (land bridge) between Nero Lake and Hansen Bay may be acting as a barrier for the contaminants in Nero Lake.

Gunnar

The Gunnar site is located 25 km southwest of Uranium City on the north shore of Lake Athabasca. A conventional acid leaching circuit was operated by Gunnar Mines Ltd. from 1955 to 1964 with the unneutralized tailings deposited into Mudford Lake. Some mine backfill of coarse tailings was also carried out at one time during the operation. A total of 5,500,000 tonnes of tailings were produced.

Mudford Lake basin eventually filled and additional tailings flowed north by gravity through 2 small lakes and into Langley Bay of Lake Athabasca. Studies in this area have indicated the following site conditions:

- the tailings contain very little pyrite with the pH being neutral or slightly alkaline.
- the majority of the tailings are in the Gunnar Main area. These tailings are mostly dry with the upper portion subject to wind erosion.
- the "Central A" tailings are overgrown and the "Central B" tailings are dry and overgrown.
- the tailings delta in Langley Bay consists of fine tailings in the lake which closes off an arm of Langley Bay.
- the main factors limiting the spread of vegetation is surface instability and lack of moisture.
- the Langley Bay tailings are contributing to suspended solids, dissolved solids and radionuclides in Lake Athabasca (12-16pCi/L Ra226 in the bay in 1978).

A condition site-specific to the Gunnar area is the flooded open-pit and the waste rock piles that are present. The open-pit and underground workings were flooded by Lake Athabasca and in 1966 the open channel was closed leaving an isolated,flooded pit.

The following conditions have been identified in this flooded pit:

- there are five different strata in the 110 m deep open-pit.
- concentration of radionuclides are 5 to 7 times higher in the bottom-stratum levels below the chemocline.
- uranium in the sediment is 2 to 3 orders of magnitude greater than lake bottoms in the area while radium is 2 times greater.
- water quality in the upper levels is satisfactory.
- fish caught in the open pit contain radionuclide levels of 1 to 2 orders of magnitude greater than background.
- there is no surface seepage out of the open-pit.
- the waste rock piles are contributing contaminated seepage to Lake Athabasca.

Port Radium

The Port Radium site, mined for uranium by Eldorado Nuclear Limited, is located in the N.W.T. on the shores of Great Bear Lake approximately 400 km northwest of Yellowknife.

In the early years of operation radium and silver were recovered by gravity concentration and the tailings were discharged to the Cobalt Channel in Great Bear Lake.

Uranium recovery by acid leaching was initiated in 1942 and as part of this operation tailings in the Cobalt Channel were reprocessed for uranium recovery and subsequently discharged into small lakes on the peninsula and into Murphy Bay.

From 1964 to 1982 Echo Bay Mines Limited operated a silver mill in Port Radium using some of the existing facilities on site.

Due to the presence of some tailings at the bottom of Great Bear Lake in Cobalt Channel and Murphy Bay and due to the acidic nature of the uranium tailings that have been identified,further investigations are likely warranted for this area. This would provide an assessment of any potential environmental effects from these wastes.

Rayrock

The Rayrock tailings site is located approximately 150 km northwest of Yellowknife in the N.W.T. The mill was operated from 1957 to 1959 by Rayrock Mines Ltd. The operation generated approximately 80,000 tonnes of tailings which were deposited in two separate areas, a small wet swampy area adjacent to Sherman Lake, separated by a raodway of waste rock, and a smaller area approximately 500 m away near a small lake. The two areas of tailings are in separate drainage systems.

Although some environmental sampling has been carried out in the area, results were inconclusive and a survey of existing environmental conditions at this site would seem warranted.

Uranium milling activities are thought to also have taken place near Hettah Lake (N.W.T.) by United Uranium Corporation Ltd. in 1952 and by International Uranium Mining Corporation near Contact Lake (N.W.T.) from 1936 to 1938 but little information is available on these properties.

Bicroft

The Bicroft tailings sites, located near Bancroft, Ontario, are made up of three tailings area; Auger Lake, Bicroft Proper and the Bicroft Swamp.

The mill, operated by Bicroft Uranium Mines Ltd., used an acid leach process and discharged approximately 2,000,000 tonnes of neutralized tailings from 1958 to 1963.

Some general comments on the site conditions at Bicroft include:

- the Auger Lake and Bicroft Proper areas have revegetated with indigenous vegetation.
- beaver activity results in flooding of the Bicroft Swamp area but some revegetation has taken place.
- pH of water on the tailings surface is in the range of 6 to 7.
- water quality on the tailings surface is poor but the water quality in adjacent creeks and lakes is good.

Faraday

The Faraday Mine was operated by Faraday Uranium Mines Ltd. from 1957 to 1964. A total of 1,800,000 tonnes of tailings were deposited in a swampy area near Bentley Lake.

These tailings, also known as Madawaska Tailings Pond No. 2, were covered with overburden, seeded and fertilized to provide a good cover of indigenous vegetation.

Nordic

The Nordic mine, located approximately 6 km south of Elliot Lake, Ontario, was operated by Rio Algom Limited from 1957 to 1968. Approximately 11,000,000 tonnes of neutralized tailings were discharged to the tailings area which is contained by rockfill dams on three sides and a hill on the north side. The mill circuit consisted of acid leaching, ion exchange and ammonia precipitation, which is the typical uranium circuit operated in the Elliot Lake area.

The complete tailings area was revegetated by Rio Algom Limited during the period 1971 to 1978. The complete area is well revegetated with trees naturally encroaching on the surface. Wind erosion problems have virtually been eliminated.

The acidic seepage and runoff from the Nordic tailings is collected and treated with lime and $BaCl_2$ for neutralization and radium removal. North Nordic Lake is used as a settling pond for the waste sludge.

Investigations at this site have provided the following information:

- the tailings are pyritic (5-6% FeS_2) and acid generating.
- primary concern is the acidic seepage with its inherent heavy metals content and radionuclides.
- the front zone of low pH seepage water moves at a slower rate than the groundwater front due to neutralization effects of calcite in the sand aquifer.
- in front of the low pH zone the concentrations of heavy metals and radionuclides are acceptable.
- behind this front the pH is low and high concentrations of heavy metals and radionuclides exist.
- lowering of the water table to assist in revegatation of slimes area resulted in an increase in depth of pyrite oxidation and acid generation.
- precipitation and adsorption reactions in the pH control zone plays an important part in the control of contaminants in the groundwater.
- the vegetated cover does not seem to reduce the rate of pyrite oxidation in the shallow zone or reduce the net downward movement of imput water.

Lacnor

The Lacnor mine, located 6 km from the town of Elliot Lake, was operated by Northspan Uranium Mines Limited from 1957 to 1960. A total of approximately 2,700,000 tonnes of tailings were neutralized and deposited into a small lake.

The tailings are well covered with introduced vegetation but some bare spots remain due to surface drainage problem. A portion of the tailings are underwater.

The seepage and runoff from this tailings site is directed, by gravity, to the Nordic tailings area and the contaminated water is combined with the Nordic seepage and treated in the Nordic treatment facility.

Stanrock

The Stanrock operation, located north of Elliot Lake, Ontario, was operated by Stanrock Uranium Mines Ltd. from 1958 to 1964 with approximately 5,800,00 tonnes of neutralized tailings being discharged to the tailings site. An additional 2,200,000 tonnes of tailings were discharged to the Stanrock tailings site from the adjacent Can-Met mill operated by Can-Met Exploration Ltd. from 1957 to 1960.

The Stanrock operation continued to recover uranium with a bacterial leach operation from 1964 to 1969.

One dam, constructed of classified tailings sands, at this tailings area failed and caused tailings to spread into an adjacent swampy area and creek.

The Stanrock main area was sprayed with lime in 1978 as a method of dust control. This has encouraged the encroachment of natural vegetation onto the tailings area. Seepage and runoff from this tailings site is collected and treated with lime and $BaCl_2$.

Spanish American (Olive Lake)

The Spanish American mill was operated for a brief period by the Northspan group in 1958-1959. A total of 450,000 tonnes of neutralized tailings were discharged to Olive Lake. The majority of the surface tailings are revegetated and some of the tailings are underwater. The seepage from this area flows by gravity to the Long Lake tailings area where it is treated with the Denison tailings seepage and runoff.

Pronto

The Pronto mine, located near Spragge, Ontario on Highway 17, was operated by Pronto Uranium Mines Ltd. from 1955 to 1960. A total of 2,100,000 tonnes of tailings were discharged to the tailings site.

The uranium tailings have been partially revegetated but copper tailings, from a period of copper concentrate production at this mine, are not revegetated due to their compaction and slimes content. Surface runoff (pH 3) is treated with $BaCl_2$ followed by lime neutralization to pH 8-9.

CLOSE-OUT TAILINGS

At present there are 3 operations that are currently in a close-out situation. They include the Beaverlodge operation in Uranium City, Sask., Madawaska Mine in Bancroft, Ont. and the Agnew Lake Mine near Espanola, Ont.

Beaverlodge

The Beaverlodge operation of Eldorado Nuclear Limited, located on the shore of Beaverldoge Lake in northern Saskatchewan, was shut down in June 1982 after 30 years of operation. The mill feed originated from the Fay/Verna Mine, the Dubyna Mine as well as many satellite mines in the area. The mill circuit used a carbonate leaching circuit and caustic precipitation. The tailings from this operation are alkaline and are not acid generating.

Mine backfill was practiced in this operation with approximately 3,500,000 tonnes of tailings used for this purpose. The remaining tailings, approximately 6,000,000 tonnes, were deposited in nearby lakes.

The tailings decant water is treated with barium chloride, followed by settling in settling ponds prior to discharge to Beaverlodge Lake.

Water cover of tailings areas is the present conceptual close-out plan for this area but other site-specific options are currently being evaluated.

Madawaska

The Madawaska operation near Bancroft, Ontario is presently on a care-and-maintenance basis with the tailings area being reclaimed with a rip-rap cover of waste rock and gravel.

The operation was re-activated by Madawaska Mines Ltd. in 1977 and operated until June 1982 using an acid-leach circuit, ion exchange and magnesium precipitation. A total of approximately 2,000,000 tonnes of neutralized tailings were deposited to the tailings site east of the mill during this period. Approximately half of the tailings decant water was re-used in the mill and the remainder was treated with $BaCl_2$ and discharged via a settling pond and gravel bed.

Site specific comments for this tailings area are:

- the gravel bed for effluent clarification is acting as a radium sink.
- the tailings are not acid-generating and thus contaminant migration problems should be minimal.
- the base beneath the tailings area has good cation exchange capacity which will ensure the capture of any contaminant that may be contained in the seepage.
- the grout curtain that was installed led to a reduction in seepage flow through the east dam but it was not entirely cut off and some seepage occurs into Bentley Lake. However, the cation adsorption capacity in the area reduces the radium to acceptable levels.

Agnew Lake

The Agnew Lake mine, located near Espanola, Ontario, is operated by Agnew Lake Mines Ltd. using underground leaching and heap leaching with ion exchange/solvent extraction and ammonia precipitation. This operation which started up in 1977, is due to close in 1983 and reclamation procedures have already been initiated.

The tailings from this operation consist of precipitate wastes resulting from the neutralization and treatment of waste solutions from the plant. The wastes are discharged to a valley which is isolated by an impervious, hypalon-lined dam on one side and an earth-wall dam with a control tower at the other end. Seasonal discharge of effluent to a polishing pond and then to the water course is practiced. The other waste material from this operation includes approximately 2 million tonnes of leached ore on surface.

The close-out plan currently being carried out includes the transfer of the leached ore to the sludge pond which will reduce the acid-generating potential of the ore while allowing the sludge pond to be solidified and revegetated. Vegetation tests have shown that a sand cover over the waste area would be sufficient to provide the necessary vegetation cover.

ACTIVE TAILINGS

The presently active tailings areas in Canada include the facilities for Quirke, Panel and Denison in the Elliot Lake area and the Cluff Mining and Rabbit Lake operations in northern Saskatchewan. The capacities of each mill are given in Table 2.

TABLE 2

OPERATING URANIUM MILLS

Mine	Capacity (t/day)
Quirke	6400
Panel	3000
Denison	11000
Rabbit Lake	1800
Cluff Mining-Phase 1	*

* presently reprocessing gravity tails from Phase 1 operation

Quirke

The Quirke mine and mill, operated by Rio Algom Limited, was operated from 1956 to 1961 and then re-started in 1968. The mill, using acid leach, ion exchange and ammonia precipitation circuits has a capacity of 6400 t/day but is currently milling at a rate of 4500 t/day.

The neutralized tailings are discharged as a slurry to the Bud Lake tailings basis which presently contains approximately 27,000,000 tonnes of tailings in an approximate area of 80 ha. This tailings area is presently being expanded to provide additional storage capacity which is required for the continuing operation.

The tailings decant water is treated with $BaCl_2$ in a series of three settling ponds to reduce the radium to acceptable levels prior to discharge to the Serpent River. The build-up of sludge in these ponds is a problem which is being addressed at the present time.

Panel

The Panel operation, operated by Rio Algom Limited, was active from 1958 to 1961 and was re-activated in 1979. The mill process includes acid leaching, ion exchange and magnesia precipitation with a capacity of 3000 t/day. Tailings from the previous operation are estimated at 3,300,000 tonnes and new tailings deposition is approximately 3,000,000 tonnes to date.

The neutralized tailings are discharged to the Strike Lake tailings management area and are deposited over a portion of the old tailings.

Tailings decant water is collected and treated with $BaCl_2$ and lime for radium removal and neutralization. Effluent is clarified in settling ponds prior to discharge to Quirke Lake.

Hypalon liners have been used in the construction of the perimeter tailings dams and for the baffled, settling ponds used for treated effluent clarification.

In the design of the expanded tailings area for the Panel mine reactivation a small area of old tailings located east of Dam A was left abandoned and remained in the Rochester Creek watershed. Studies had shown that this small area of tailings had no impact on the environment since the water flow in Rochester Creek is sufficiently high to mask any possible contaminant problem from these tailings. Furthermore, as most of the tailings are underwater acid generation, and hence contaminant migration, does not appear to be a problem.

Denison

The Denison mine, operated by Denison Mines Limited, has been in continous operation since 1957. The mill throughput has been increased in stages to the present capacity of 11,000 t/day. The mill circuit is similar to the Quirke operation, that is, acid leaching, ion exchange and ammonia precipitation.

A total of approximately 40,000,000 tonnes of tailings has been deposited in the Williams Lake and Long Lake tailings basin. The Williams Lake area is no longer used and some reclamation work has been carried out in the lower Williams Lake tailings area using a glacial till cover and vegetation with grasses and legumes. The upper Williams Lake area contains exposed tailings.

The Long Lake basin is made up of the main dam at the west end and a dam at the east end. Tailings slurry is discharged at the east end which, in practice, results in the segregation of coarse and slimes with the slimes reporting to the west end of the tailings area.

Tailings decant is treated with $BaCl_2$ and discharged to Stollery Lake which is separated by a dyke to allow settling of the precipitated sludge before disposal to the Serpent River system.

Cluff Lake

The Cluff Lake operation in northern Saskatchewan, operated by Amok Ltd., consists of 2 separate operating phases.

The Phase 1 operation, started in 1981, included the treatment of the very high-grade "D" orebody with a milling process consisting of gravity concentation, acid leaching, and direct-precipitation of the leach liquor with magnesia. This operating phase was completed in early 1983 at which time the re-processing of gravimetric residues was initiated as an extension of the Phase 1 operation. This will continue to operate until 1984.

The Phase 2 operation which is currently being planned, and due to start in late 1984, will be recovering uranium from lower grade (0.5% U_3O_8) ore deposits in the area using a conventional acid leach, CCD and solvent extraction circuit followed by magnesia precipitation in the existing facilities. The planned throughput is 230,000 tonnes/year.

The wastes from the Phase 1 operation included the leach residues, which are presently contained in concrete bins stored under cover on surface, and waste solutions containing a minimal amount of precipitates.

The waste solutions from Phase 1, as well as minewater, reported to the tailings containment area. Decant solution is treated in the secondary treatment system (STS) with $BaCl_2$ and a polymer. The treated effluent is clarified in settling ponds and dual-media sand filters prior to discharge to the environment.

Solid tailings from the gravimetric residue reprocessing phase will be neutralized and discharged to the present waste management area. Decant solution will continue to be treated in the STS.

The present proposal for tailings impoundment during the Phase 2 operation is to use the existing waste management area and to increase the height of the Phase 1 dam to provide the required storage capacity. Decant solutions, and excess minewater, would continue to be treated in the STS. Although this is a viable placement method other alternatives are currently being contemplated. These include the use of a cone, or flattened-cone, placement method in the existing facility or temporary storage in the existing facility followed by disposal of the tailings slurry in the Claude open-pit once mining activities are completed for the Claude orebody. Tailings in the existing area would then be transferred to the Claude pit at the appropriate time. Decant solution would continue to be treated in the STS. Close-out modelling studies have been commissioned for these alternatives.

Rabbit Lake

The Rabbit Lake operation, located on the west side of Wollaston Lake in northern Saskatchewan, was started-up in 1975 by Gulf Minerals and taken over by Eldor Mines Ltd. in late 1982.

The current milling process uses an acid-leach circuit, CCD's, and solvent extraction followed by ammonia precipitation. The plant is operated at 1500 to 2000 t/day with neutralized tailings discharged to a tailings basin constructed by damming a large natural depression. Approximately 3,500,000 tonnes of tailings have been discharged to date.

The tailings decant water is treated with $BaCl_2$ and polymer to remove radium and clarified in a series of three settling ponds prior to discharge to Hidden Bay. A sand filter has been installed and a portion of the treated effluent is clarified in this circuit.

Although heavy metal and radionuclide levels have been acceptable with the current treatment system, elevated ammonia levels in the effluent have presented problems in the past. The use of ammonia will no longer be required in the proposed circuit for the Collins Bay ore and thus future problems are not anticipated.

NEW TAILINGS SITES

New uranium operations which are committed to start up in the near future include the Key Lake Project in northern Saskatchewan, with a planned start-up in the latter half of 1983, and the reactivated Stanleigh Mine in Elliot Lake, Ontario with a start-up date of July 1983. The Eldor Mines Ltd. Collins Bay Project, a continuing phase of the existing Rabbit Lake operation, is slated to start-up in 1984 and the Phase 2 stage of the Cluff Mining operations is due for start-up in late 1984. A list of the new projects with their respective mill capacities is given in Table 3.

TABLE 3

NEW URANIUM MILLS

Mill	Capacity (t/day)
Key Lake	720
Stanleigh	4250
Collins Bay Project	1800
Cluff Mining - Phase 2	790

Key Lake

The Key Lake Project, operated by Key Lake Mining Corporation, is located approximately 200 km north of LaRonge, Sask. The ore is a high-grade (2.5% U_3O_8) uranium ore containing high levels of heavy metals and arsenic as contaminants. Radionuclides consist of radium and uranium with very low thorium levels.

The milling circuit, with a capacity of approximately 700 tonnes per day, consists of a 2-stage acid leach with CCD thickeners and solvent extraction for uranium recovery and purification. Uranium will be precipitated with ammonia and an ammonium sulphate crystallization plant will ensure acceptable ammonia levels in the final effluent.

Leach residues will be neutralized with lime and discharged to the tailings area with waste precipitates from the separate waste solution treatment circuit.

Treated effluent will be clarified using clarifying thickeners and pre-coat filters and discharged to monitoring ponds for check-analysis prior to being discharged to the environment. This mechanical separation of waste precipitates from treated effluents should prove to be much more efficient than settling ponds.

The bulk tailings will be pumped to a tailings containment area as a 35-40% solids slurry. The tailings area is being constructed to accomodate a layered tailings deposition technique. The construction basically consists of a bentonite seal, underdrains, and a sand filter bed for collection of seepage and runoff during the deposition phase. Tailings will be systematically placed in layers, approximately 100 mm in depth, and allowed to consolidate by evaporation and drainage. In cold weather, end discharge will be practiced in selected sections of the tailings site. These thicker layers will be allowed to consolidate over a certain time period prior to the placement of further tailings. The discontinuity effect of these thicker layers on the system is not known and will need to be determined.

The concept of this tailings placement method is to have a potential "walk-away" tailings containment area upon decommissioning. The systematic layered deposition method is to provide a partially saturated, moisture-retaining, tailings mass of low permeability which will result in negligible seepage after a few years of operation. On close-out, treatment of seepage and runoff water would therefore not be required.

Although operating problems can be anticipated with this type of placement method, especially under cold weather operations, the concept is certainly acceptable in light of its potential as a walk-away situation.

Collins Bay

The Collins Bay B-Zone orebody will be milled in the existing Rabbit Lake mill with some circuit modifications to accomodate the varying mineralogy of this ore compared to the Rabbit Lake ore. The main differences of this ore are its arsenic content as well as other minor constituents which could affect the quality of the final uranium product. Changes to the Rabbit Lake circuit will include, the use of strong-acid stripping for solvent extraction, the use of hydrogen peroxide for uranium precipitation and modifications to the waste treatment circuit for efficient removal of contaminants from minewater and mill effluents.

The tailings disposal method which has been approved for this operation is in-pit disposal using the old Rabbit Lake open-pit to contain all the tailings from milling Collins Bay B-Zone ore.

The tailings will be dewatered by filtration in the mill in order to accelerate the consolidation of tailings when placed in the pit. The pit envelope will be pervious on all sides and at the bottom. The remaining tailings water from tailings consolidation in the pit will be collected at the base of the pit and pumped via a raise to surface and then to the effluent treatment plant. Sand filters will be used for clarification of final effluents.

During the operating phase, the water table will be kept below the bottom of the pit (by pumping of seepage and groundwater) to allow consolidation of tailings, and on close-out the surface of the tailings will be covered with a soil cover and the pit area allowed to return to its natural state, Rabbit Lake. Thus, the tailings mass will be isolated beneath Rabbit Lake and groundwater will flow around the tailings due to a lowering of the hydraulic gradient caused by the pervious envelope around the consolidated tailings mass.

Stanleigh

The Stanleigh Mine, located in Elliot Lake, Ontario, is presently being re-activated by Rio Algom Limited with a planned start-up date of July 1983.

The Stanleigh mill was previously operated from 1958 to 1960 and the tailings were discharged to the Crotch Lake tailings area along with tailings from the Milliken operation which ran from 1958 to 1964. A total of approximately 7,500,000 tonnes of tailings were deposited from these 2 operations.

The new Stanleigh mill operation will have a capacity of 4250 t/day with neutralized tailings being pumped to an expanded Crotch Lake tailings area. Tailings area decant water, which is acidic, will be treated with $BaCl_2$ and lime and clarified with dual media sand filters in a new treatment plant located at the discharge end of Crotch Lake. Sand filter backwash, containing the radium sludge, will be returned to the tailings basin by pumping underwater towards the area of tailings deposition. This method will allow the radium sludge to mix with encroaching tailings during settling. The use of sand filters for clarification should prove to be much more efficient than the conventional settling ponds that have been used to date in the Elliot Lake area.

Cluff Mining - Phase 2

The Phase 2 operation of Cluff Mining is due to start-up in late 1984 at a capacity of 790 t/day. Although a general description of this project has been given above final plans have not been competed with respect to tailings containment.

EFFLUENT TREATMENT

The requirement of waste treatment in uranium operations include the neutralization of residue, treatment of mill waste solutions, treatment of contaminated minewater and treatment of tailings seepage and runoff at active and inactive tailings sites.

Effluent treatment can either be carried out in the mill as part of the complete mill process or at the tailings management area where secondary treatment of the tailings supernatant or seepage water is carried out for radium removal.

Both methods are used in northern Saskatchewan. At Rabbit Lake, leach residue tailings and waste solutions are bulk-neutralized with lime in the mill and the combined tailings slurry is pumped to the tailings containment area. Tailings water is then treated with barium chloride and coagulant for radium removal. Settling ponds are being used for effluent clarification but a sand filter unit is also used for clarification of part of the flow. The Collins Bay Project will include sand filters in the design for positive removal of radium precipitates from treated effluent water.

The Cluff Lake operation is using a similar system of bulk neutralization for the leach residue and raffinate solution. Tailings water is treated in a secondary treatment system with barium chloride and alum. Radium sludge removal is accomplished in a settling pond followed by sand filters which were installed during the operating phase of the project. Excellent radium levels are being achieved for the final effluent.

The Key Lake Project will be neutralizing waste solutions and treating for radium removal as part of the mill process. Removal of precipitates from treated solutions will be accomplished mechanically by the use of clarifying thickeners and filters. These waste products, containing calcium sulphates and arsenates, iron hydroxides, heavy metal precipitates and radium precipitates, will be combined with the neutralized leach residue for disposal as a mixture in the tailings area. Clarified effluent will be collected in monitoring ponds for check analysis prior to release to the environment. This system allows the contaminant precipitates and solid tailings to be mixed prior to disposal and therefore eliminates the problem of waste sludge build-up in settling ponds. The use of filters for final clarification can also be expected to be much more efficient than settling ponds for removal of the fine precipitates that are produced in waste treatment circuits. The proposed circuit for the Midwest Project also includes mechanical thickening and filtration as part of the design for waste solution treatment.

At Madawaska, excess tailings water which was not re-used in the process was treated for radium removal by the addition of barium chloride and clarification was carried out in a settling pond followed by a gravel bed where the treated effluent discharged to the groundwater. This system is unique in that the gravel bed not only clarifies the effluent but also acts as a collector for radium due to its inherent adsorption characteristics.

In Elliot Lake the existing operations (Quirke, Panel, Denison) bulk-neutralize waste materials in the mill and treat tailings decant water at the tailings site for radium removal. Although regulatory radium levels are currently being met the build-up of waste sludge in the settling ponds used for clarification could present problems in the near future.

The Panel operation is using hypalon-lined, baffled, settling ponds. Good results are being achieved but sludge build-up is becoming evident in the front sections of the ponds.

The Stanleigh Project has included a well-engineered effluent treatment plant in the project design. Crotch Lake tailings area runoff will be treated with lime and barium chloride for neutralization and radium removal. The treated effluent will be clarified in dual-media sand filters and sand filter backwash, containing the fine precipitates, will be pumped back underwater into the tailings area. The encroaching tailings will settle over the precipitate sludge thereby assisting in settling and immobilization of the wastes.

The treatment of contaminated seepage and runoff at Beaverlodge and at inactive tailings sites in the Elliot Lake area is resulting in the build-up of waste sludge in dedicated lakes or ponds. This can be considered as one of the major problems which will need to be addressed for the long-term close-out of these inactive sites.

In summary, the use of mechanical equipment for liquid-solid separation and clarification of treated effluents will likely become common place in future uranium plant designs. This will allow the physical immobilization of the waste sludges within the solid tailings matrix and also provide excellent clarification of treated effluents prior to release to the environment. Problems which still need to be addressed include the removal and permanent disposal of waste sludges in the settling ponds at active and inactive tailings sites as well as the potential re-dissolution of contaminants in these sludges after disposal.

IMPOUNDMENT TECHNIQUES

The method of tailings disposal varies from site to site and depends on the site-specific nature of the local area. The two broad categories of tailings containment include surface disposal which require the construction of engineered embankments on surface and sub-surface disposal where tailings are placed below ground. The various containment methods are briefly described below.

Surface Disposal

Placement methods for surface impoundments areas include the wet discharge method, as practiced in the operations at Elliot Lake, Ontario, thickened tailings discharge, layered tailings deposition and filtered tailings discharge.

i) Wet Discharge

In the wet discharge method neutralized tailings pulp from the mill is pumped to the tailings area where lateral segregation of coarse and fines occurs from the point of discharge. The slimes settle out and clear decant water is removed for treatment and discharge. A single spigot or multiple spigots can be used for this application.

This method results in a substantially saturated tailings deposit with zones of unconsolidated slimes. The high water table provides a high pressure head which increases the rate of seepage from the tailings area. On close-out the large areas of slimes that are present can lead to difficulties in reclamation.

ii) Thickened Tailings

The cone discharge of thickened tailings is being evaluated at Denison in order to increase the capacity of the Long Lake tailings basin. In addition to resulting in an increase in storage capacity, the final contour of the tailings surface will be better suited for reclamation at close-out. Also, the continuous slope which develops with this method of discharge is suited to shedding of surface water rather than infiltration and should therefore results in less seepage.

iii) Layered Tailings

The layered deposition of tailings will be used at the Key Lake operation as described above.

The deposition of tailings in thin layers with a shallow slope results in vertical classification of sands and slimes in each layer. Successive layers of tailings deposited on the beaches result in a stratified deposit of tailings with low permeability and thus input water reports mostly as runoff instead of seeping through the tailings.

Consolidation of the tailings is achieved by evaporation and by removal of excess water via underdrains installed below the tailings.

The slimes that concentrate at the top of the layers will tend to form a crust on drying and thus wind erosion problems can be minimized during the drying phase. Also, seepage and runoff water can be recycled to the dam as sprays in order to increase the net evaporation of tailings water and to keep the tailings surface damp.

On decommissioning the tailings are left in a consolidated state with moisture retaining capacity.

The concept of layered tailings has been applied for many years in South Africa and the above benefits of this method have been proven. However, the severe cold northern climate typical of northern Saskatchewan will require the deposition of thicker layers during the winter months and the effect of this discontinuity on the system is not known. Other field problems that may occur include surface cracking, which may increase permeability, and the possibility of lateral segregation due to variations of tailings pulp densities. This could result in increased seepage at the sandy zones near the point of discharge.

The application of this method at Key Lake will greatly advance the state-of-the-art of this deposition technique in Canada.

iv) Filtered Tailings

The proposed in-pit tailings disposal method for the Collins Bay B-Zone Project will require filtration of the tailings in the mill prior to disposal. Filtration is carried out in order to accelerate consolidation of the tailings when placed in the pit. In turn this provides recycle water for the mill process. Collins Bay will be the first major application of filtered tailings in Canada.

Filtered tailings could also be deposited in surface impoundments in order to reduce the area requirement for tailings sites as well as providing consolidated tailings for enhancing close-out procedures.

The higher cost of filtered tailings deposition, due to trucking or conveying requirements, compared to more conventional systems precludes it from being included in current uranium plant designs. Experience from the Collins Bay Project may result in more interest for this method of deposition.

Subsurface Disposal

i) Pit

The first Canadian application of in-pit disposal of tailings in a mined-out pit will be the Collins Bay B-Zone Project in northern Saskatchewan.

The tailings originating from milling Collins Bay ore will be filtered and deposited in the mined-out Rabbit Lake pit. In order to avoid contamination of groundwater a zone of coarse rock and sand will be placed between the pit wall and the tailings thereby causing a hydraulic "short-circuit" which will prevent interactions between groundwater and tailings water. Tailings water will be collected and treated during the operating phase and for a certain period during the transition phase. Final decommissioning will allow the overlying lake basin to return to its original state as a water body.

In-pit disposal is being practiced in the United States at Union Carbide Gas Hills Plant where the pit bottom and sidewalls were lined with clay and the water originating from the hydraulically-placed tailings was reclaimed through an in-pit filter blanket and submersible pumps for pumping to lined evaporation ponds.

The main advantage to in-pit disposal results in the complete isolation of the tailings on decommissioning. Gamma radiation and radon emanation is virtually eliminated, erosion problems are non-existent and there is no need to have an on-going monitoring and maintenance program after decommissioning. Although this scheme is very site-specific (basic requirement is a mined-out pit) these advantages make it a promising disposal alternative.

ii) Mine Backfill

The use of classified mill tailings mixed with cement for use as ground support medium in cut-and-fill mining method has been practiced at Madawaska and Beaverlodge. Denison is introducing it at their operation for secondary pillar recovery. This mining method results in approximately half the mill tailings being returned underground with the remaining half, the slimes, requiring separate containment on surface.

A greater use of tailings fines in backfill operations may be possible by employing slime modification techniques such as flocculation, solidification or electrokinetic densification. Further investigations would be required to assess the suitability of these techniques.

There is some concern however that the higher levels of radium in the slimes fraction of tailings could present problems for backfill operations due to radon emanation if a greater use of the fine fraction was practiced. Also, possible contamination effects on the groundwater by backfill-placed tailings is not known and further investigations may be necessary in this area.

The placement of tailings in abandoned and exhausted mines or in mined-out stopes in active mines, could be considered for both currently-produced tailings or abandoned tailings. The technique of choice would be to attempt the separation of the fine and coarse fractions with the slimes being placed underground and the coarse, with its lower radionuclide content, stored on surface. Although this possibility may be technically feasible it would likely prove to be too costly and, more important, it would preclude future mining activities in selected mines. This would not be acceptable as market conditions and improving technologies have resulted in certain mines being re-activated after a period of dormancy.

Overall then the use of mine backfill methods should be encouraged since it reduces the total amount of tailings requiring surface containment.

iii) Deep Lake

The possibility of deep lake disposal of tailings has been considered for the Elliot Lake operations since this would eliminate the long-term problem of acid generation. Stage 1 of the Quirke Lake Uranium Tailings Disposal Study for the Elliot Lake area is currently being reviewed by the Ontario Ministry of Environment, the mine operators in Elliot Lake, Ontario and federal and provincial regulatory agencies. A water cover has been shown to retard acid generation for pyritic tailings at certain abandoned tailings sites in Elliot Lake. In addition, lakes are depositional environments and the tailings at the bottom of a lake would, with time, be covered with lake sediments. This would reduce the exchange of contaminants between the tailings mass and the overlying water. The isolation of the tailings could be further enhanced by the placement of a clay or bentonite layer over the tailings after tailings placement is completed.

In a suitable lake a stable, stagnant bottom layer of water would be isolated from the upper layers by a chemocline. This effect has been observed at the flooded Gunnar open pit in northern Saskatchewan. As a result, in a lake with downward hydraulic gradients, any potential contaminant release from the solid tailings would remain in this stagnant bottom layer.

Tailings in a deep lake disposal scheme could be considered as being completely isolated from the surface with gamma radiation and radon emanation being eliminated. Although many technical uncertainties exist for this type of disposal method, the advantages of this method over surface storage options seems to provide justification for further evaluations.

iv) Cellular

The use of specially excavated cells and trenches has been practiced in the United States for uranium tailings disposal.

The White Mesa mill in southeastern Utah features simultaneous construction, operation and reclamation of sequential cells lined with hypalon. The Sweetwater Project uses a 4-cell impoundment area lined with PVC with each cell excavated above the water table. Final reclamation includes a 0.6 m clay cap and a 3.5 m cover of local soils.

The Mt Taylor Project in New Mexico proposed the use of a series of parallel excavated trenches for hydraulic deposition of the slimes fraction of the tailings. The coarse fraction was to be used as mine backfill. The spoil from the excavated trench would be placed over the drained tailings placed in the previously filled trench. The Pioneer Project in Colorado proposes the disposal of dewatered tailings in a series of below-grade trenches.

Although these disposal options can be considered as alternatives they are costly and the site characteristics of uranium properties in Canada would preclude them from being viable options for Canadian operations.

CONCLUSIONS

The description of tailings sites given above along with details of effluent treatment practices as well as the potential impoundment techniques available for Canadian uranium operations has assisted in arriving at research recommendations for the N.U.T.P.

A description of waste disposal technologies has also been provided in the detailed report prepared for the N.U.T.P. This included such topics as site selection, liners, subsurface flow barriers, tailings modification methods, reclamation practices, waste rock aspects and the possibility of in-situ leaching as an alternative to conventional mining and milling.

An overall assessment of this information has provided some insight into potential avenues of investigation for arriving at procedures for the long-term close-out of uranium tailings. This in turn has afforded numerous research suggestions for the waste disposal technology component of the National Uranium Tailings Program and provided a basis for discussion to outline a detailed research program on disposal technology.

Acknowledgment

The author expresses his appreciation for the support received for this work from the National Uranium Tailings Program.

Discussion of Paper by L. Melis

E. Joe	Can you report on the lack of progress by Key Lake principals on tests on the tailings in the "coffins" on decontamination?
L. Melis	The concept is being evaluated for the leach residue. If the decision is made there will be a 400/1 dilution which will probably not be a problem.
E. Joe	What was the fundamental problem?
P. Fournier	Stabilization studies on the leach tails is a long study.

Tailing Disposal by the Thickened Discharge Method for Improved Economy and Environmental Control

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INTRODUCTION

Until recently tailing disposal was dismissed as a simple nuisance factor. The only concern was to dispose of the wastes at the lowest cost possible. The need to protect the environment and to provide for future rehabilitation, coupled with the energy crisis, however, has made tailing disposal a substantial component of mine development and operating costs.

The answer to reducing these high costs cannot be found in conventional disposal methods. Building bigger and better earth-compacted dams, for example, only solves part of the technical problem, and that at enormous, and sometimes prohibitive, expense.

Ideally, it would be desirable to place the tailing back into mined-out areas. However, this is not always possible. Perhaps the next best approach would be to dispose of the wastes, in a substantially 'dry' state, at 'environmentally safe' locations, possibly many miles from the mine. All water should be recycled at the plant in order to save both pumping and heat energy. Dams should be restricted in extent and height and preferably avoided altogether.

This paper describes an alternative to the conventional disposal system, known as the thickened discharge method, which makes it possible to pursue the ideal goals of the 'next best' approach. This proven new method costs considerably less to implement and operate, and also provides an environmentally more acceptable solution. The method also has the advantage that it can be used to increase substantially the capacity of existing tailing disposal areas without the requirement to raise the perimeter dams, an operation that is usually expensive. Savings and improvements for both existing and new mines in any topographical setting are apparent when the following are considered:

- Less initial capital investment is required.
- Operational costs are reduced.
- The need for high dams with their accompanying stability problems is eliminated.
- A greater volume containment is obtained for a given height of perimeter dyke.
- The slimes settlement pond and decant system are eliminated.
- Recycling of overflow water directly from the tailing thickeners, where chemically acceptable, saves pumping and heat energy.
- Pollution of the surrounding environment by seepage and dusting is reduced.
- Rehabilitation of tailing areas is greatly facilitated and hence is made less costly.

Although these items may appear idealistic to experienced tailing disposal operators, they are now being substantiated at several mine sites.

The thickened tailing disposal method was conceived by the author in 1965 and implemented for the Texasgulf Canada Limited, Kidd Creek Mine of Timmins, Ontario. This first system has been operating successfully since 1973 at a daily tailing discharge of 7500 dry tons. Since the introduction of the thickened tailing discharge method at this mine, twelve other plants have either adopted this approach, or plan to convert to it in the future. It is to be noted that the method has been approved by United States and Canadian enviornmental protection agencies.

The first publication describing this method was presented by the author in 1975 (1). The present paper discusses the most recent information on the subject, particularly with regard to the actual depositional behavior of thickened tailing. For completeness of coverage, the basic principles of the method are also reviewed.

THE THICKENED DISCHARGE METHOD

The primary aim of the thickened discharge method is to eliminate high and expensive tailing dams, whether built of tailing or of earth fill, and to eliminate the elevated 'slimes pond' and decant system. This end is attained by depositing the tailing in a cone-shaped hill -- one of the most stable and erosion-resistant structural shapes found in nature. This shape is achieved by discharging the tailing, which has been appropriately thickened, from elevated positions. Thickened tailing, discharged as a slurry, comes to rest on a slope at its natural angle of repose. A low cone-shaped hill grows below the discharge points, without need for confining dams. The preferred slope of the hill is about 6 percent, this value having been chosen to provide safe stability and good surface drainage for future vegetation. Generally, 50 percent solids is the minimum required to achieve 'slurry' consistency. Segregation by sedimentation is inhibited and all tailing particles are deposited without sorting. Experience has shown that more than 95 percent of all solids, including submicron sizes, remain within the sloping deposit. This eliminates the need for a conventional 'slimes pond' and decant system. Initially, the solids-to-water ratio is controlled at minimum slurry consistency. This permits the tailing to spread all the way to the design 'toe' of the conical deposit. As the tailing builds up, the solids-to-water ratio is increased progressively, thus steepening the cone, but maintaining its 'toe' at the same point. If full thickening is allowed from the start, the tailing deposit will build up quickly at the point of discharge. This would make it necessary to raise the tailing line ramp immediately -- an early expense that is not warranted.

The tailing cone is accompanied by a small tailing-water pond located at the lowest point of the impoundment area. By design, this pond receives both the natural and tailing run-off from the site, together with a very small fraction of the tailing fines transported by the run-off. Eventually, all run-off water reaches the tailing-water pond naturally, without the construction of special drainage systems. Natural terrain contours can often be used to advantage for this purpose.

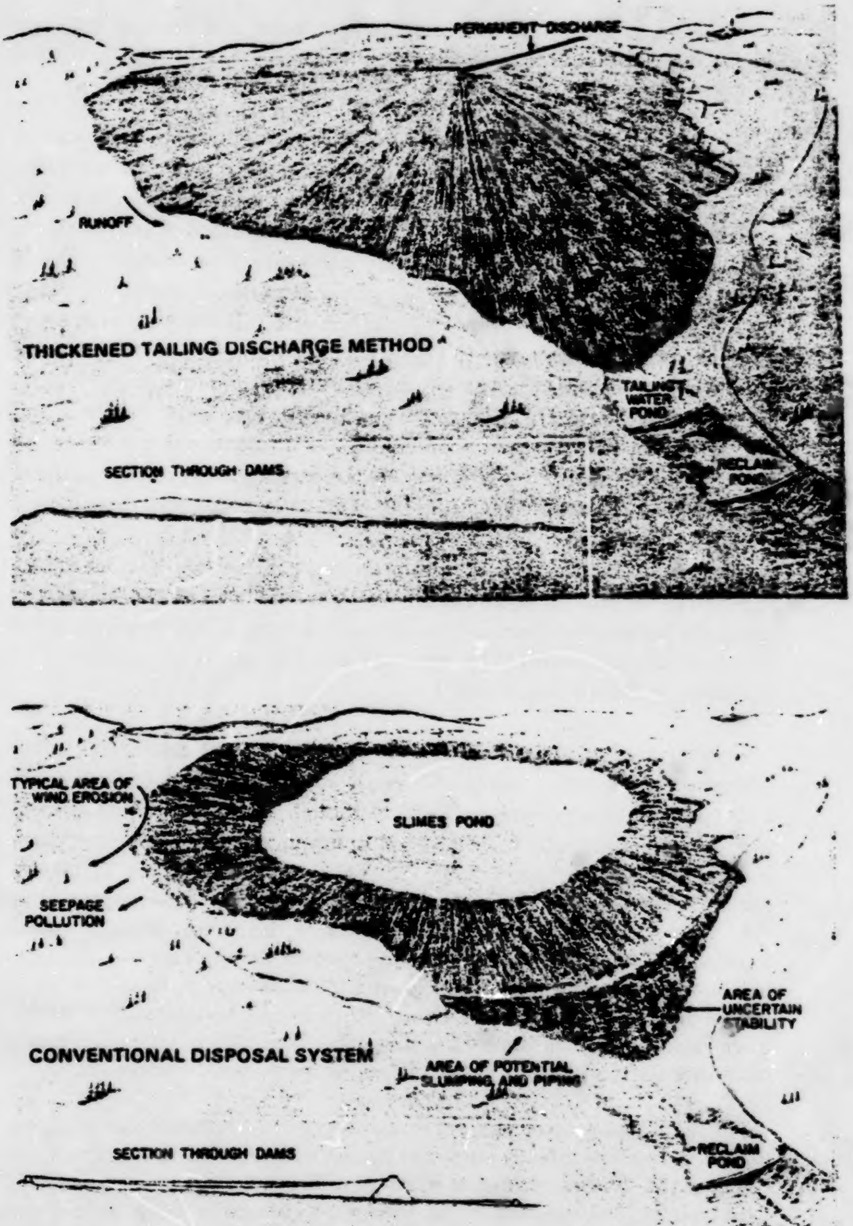
Decant towers and buried decant lines are not required. Because the pond level remains essentially unchanged throughout the life of the project, it is only necessary to supply a short overflow channel, spillway or culverts at shallow depth. A small secondary pond for chemical treatment may be established at the outfall of the overflow system prior to recycling the water.

On flat terrain, the tailing discharge point is located near the center of the area. The tailing line and discharge point are located on a ramp which is built to appropriate height. As the tailing builds up, the ramp height is increased. This is programmed to occur at two or three-year intervals. If the available area for tailing discharge happens to be oblong or rectangular in shape, then the tailing discharge is positioned initially at one end of the area. When the cone reaches its planned height, the discharge point is moved along the surface of the deposited tailing on the long axis of the area to form a ridge of tailing. In steep terrain, or valleys, the discharge points may be fixed permanently on high ground overlooking the disposal area.

Figure No. 1 shows the advantages of the method and demonstrates the adaptability of the system to any topography. A ridge-like deposit is illustrated. Discharge would have commenced at the far end of the disposal area. Only a small dam is required to contain the effluent and the small quantity of fines washed out by the run-off. It should be noted that the conical shape of the deposit tends to disperse the run-off, thus minimizing its erosive capability.

CAPITAL AND OPERATIONAL COST SAVINGS

The thickened discharge method requires only a thickener, a ramp for the discharge line in some cases, and a small permanent dyke for the tailing-water pond. The initial capital outlay, therefore, is low. This is to be compared to the conventional method which requires high dams, now commonly required to be built to


PERMANENT DISCHARGE
RUNOFF
THICKENED TAILING DISCHARGE METHOD
TAILINGS WATER POND
RECLAIM POND
SECTION THROUGH DAMS
TYPICAL AREA OF WIND EROSION
SLIMES POND
SEEPAGE POLLUTION
AREA OF UNCERTAIN STABILITY
CONVENTIONAL DISPOSAL SYSTEM
AREA OF POTENTIAL SLUMPING AND PIPING
SECTION THROUGH DAMS
RECLAIM POND

Comparison of the Thickened Tailing Discharge Method and Conventional Disposal Systems
Figure No. 1

water-retention specifications. Alternately, conventional methods require extensive starter dams around the perimeter of the area, and subsequent labour-intensive dam-building programs entailing continuing high operating costs.

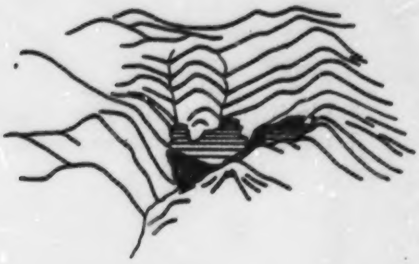
A recent study for a new mine (5000 tons per day), in an area with very strict environmental protection regulations, highlights the savings. The study showed that the disposal of tailing by the thickened discharge method would have cost 24¢ per ton compared to the existing conventional system which costs approximately 80¢ per ton - a saving of 1.02 million dollars per year. Furthermore, the thickened discharge method, will extend the life of this disposal area from 12 years to more than 70 years. After 12 years, the capital cost will have been amortized fully. Therefore, for the balance of 58 years, only operation, maintenance and occasional component replacement costs of the thickening and discharge system will be incurred. These costs are calculated to be 8 to 10¢ per ton. It should be emphasized that, after the first 12 years of operation, the conventional method becomes even more costly because the costs must be increased to cover development of new disposal areas located farther from the plant in less economical settings.

The conical-shaped deposit resulting from the thickened discharge method can be superimposed readily onto an existing conventional horizontal disposal area. Such a move will increase substantially the disposal capacity of the area without raising the perimeter dykes or dams. The approach is viable in any topographical setting, as illustrated in Figure Nos. 2 and 3.

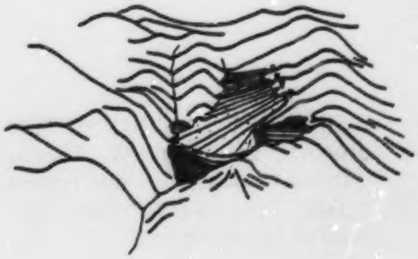
Another area of savings, both in capital expenditure and energy conservation, relates to the volume of tailing and return water to be handled. Both pipeline sizes, and pumping energy, are reduced. Although thickened tailing is heavier and therefore more difficult to pump than conventional tailing (2), the much smaller volume being handled more than compensates for this situation. Table No. I shows comparative values for a mine handling approximately 7 million tons of tailing per year. With the thickened tailing disposal system, the percent solids will eventually be increased from 60 to 65 percent and possible even higher. This will result in a further reduction in pumping energy. The lower capital and operating cost of the thickened system may be considered an inducement to locate the tailing disposal system in a more remote but environmentally perferable area. Many mines are now pumping heavy ore slurries at 60 to 70 percent solids over great distances (2).

Finally, it should be recognized that wherever it is chemically acceptable to recycle thickener overflow directly back into the plant, a substantial amount of heat energy can be saved, at least during the winter months. In the example quoted in Table No. I, a saving of 7800 kilowatts in heat-energy may be had by using the thickened tailing disposal method. For this calculation, it is assumed that the winter temperature of the normal plant water at intake is 32 degrees Fahrenheit and the tailing is discharged at 42 degrees Fahrenheit. The quantity of water not having to undergo this temperature change is the water that would be recycled directly from the thickener overflow, that is, 6410 – 1065 = 5345 U.S. gallons per minute.

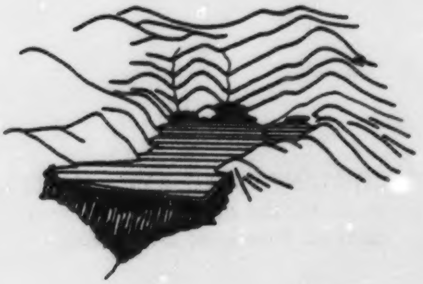
EXISTING POND will reach limiting capacity of existing 90-feet high dam in the year 1979.



THICKENED TAILING SYSTEM provides an additional 20-year disposal capacity without raising existing dam. Six percent tailing slopes are planned. The fixed discharge points are high along valley wall.



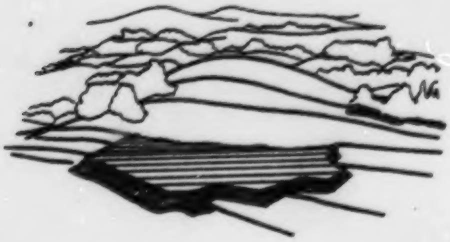
CONVENTIONAL EXPANSION to provide an additional 20-year disposal capacity would require that the existing dam be raised to a height of 225 feet. Natural construction materials must be used.



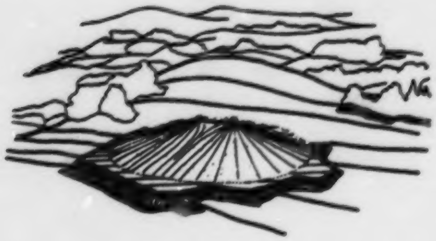
Tailing disposal at the Superior Division operation of the Magma Copper Company, Arizona.

Figure No. 2

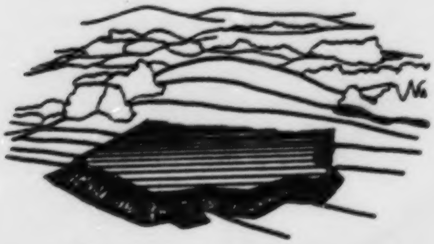
EXISTING POND will reach limiting capacity in the year 1978.



THICKENED TAILING SYSTEM provides an additional 15-year disposal capacity with five percent tailing slopes. No additional dam construction is required.



CONVENTIONAL EXPANSION would require raising the perimeter dams on three sides of the pond.



Tailing disposal at the Consolidated Durham Mines and Resources Limited operation, New Brunswick.

Figure No. 3

Table No. I
Comparison of the Thickened Tailing and Conventional Disposal Systems.
Pipeline Sizes and Pumping Energy Requirements*

Feature	Conventional Disposal System (CS)	Thickened Tailing Disposal System (TS)	Percent Reduction using TS System (CS-TS) (100)/CS
Tailing solids percent by weight	30	60	–
Discharge to tailing area - USGPM	8585	3240	62
Return water for recycling - USGPM	6410	1065	83
Total liquid to be handled - USGPM	14,999	4305	71
Tailing pipeline, inside diameter and (nominal pipe size) - inches	24 (28)	15 (18)	37
Return water pipeline, inside diameter and (nominal pipe size) - inches	17 (20)	7 (8)	59
Weight (cost) of both pipes in pounds per foot (Dupont Sclairpipe Series #100)	101.3	33.5	67
Net horsepower per mile of pipeline for pumping tailing slurry	48	47	2
Net horsepower per 100 feet elevation head for pumping tailing slurry	270	135	50
Net horsepower per mile of pipeline for pumping clean return water	106	41	61

*Assumptions: Tailing discharge is 800 dry tons per hour.
Specific gravity of solids is 2.9.
Pipeline velocities: slurry at 6 feet per second, return water at 9 feet per second.
Tailing remains at 75 percent solids in the disposal area.

SAFETY CONSIDERATIONS

The mining industry is well aware of the inherent problems associated with the operation of conventional tailing disposal schemes (3). Some of the more obvious ones are:

- Danger of foundation or slope failures in dams built of tailing.
- Piping through the dams under high hydrostatic head induced by seepage from the raised slimes pond in the center of the tailing deposit.
- Danger of liquefaction of steep-sided tailing dams caused by earth tremors, blasting vibrations, or movement of heavy equipment.
- Danger of collapse of decant towers and buried decant lines, with the inherent inaccessibility for examination and repair of the structures.
- Erosion of downstream tailing surfaces caused by natural run-off over steep slopes of tailing dams.

By the simple expedient of depositing a low conical hill or ridge of thickened tailing, the above dangers are eliminated. There are no high perimeter dams, slimes ponds or decant systems. The low profile of the developed hill ensures stability and reduces the dangers of liquefaction.

With regard to the latter, it should be remembered that the tailing slurry is discharged in a completely liquefied state. It comes to rest at a slope determined by its angle of internal friction (viscosity). Thereafter, by consolidation under the action of its own self-weight, the water content is reduced progressively until the material reaches a value of 70 to 80 percent solids. At this water content, the angle of internal friction is expected to become at least two times as high as at the originally deposited 6 percent slope. This increase in friction is shown in subsequent graphs. It is therefore concluded that a large reserve angle of friction is available to oppose the movement of tailing during earthquakes. Laboratory tests are now in progress to verify this conclusion for areas subject to earthquakes.

CONTROL OF ENVIRONMENTAL POLLUTION

Pollution from tailing disposal areas occurs in two ways:

- Uncontrolled escape of effluent and solids by seepage and run-off, and
- Wind erosion of tailing solids from dam surfaces.

Using the thickened discharge method, the point of concern regarding seepage and run-off pollution is now isolated to the small tailing-water pond below the tailing deposit. By design, all water ultimately reaches this pond. This is unlike traditional tailing systems, in which run-off and seepage from the outside surfaces of the perimeter dams can often pollute the surrounding terrain. In the tailing-water pond for the thickened discharge method any transported solids are allowed to settle out, and the pond water overflows into a reclaim pond through a spillway or culvert installed at shallow depth. The small dams required for these ponds and the shallow overflow culverts are subject to low earth pressure, simple inspection and easy replacement, should replacement be required.

Dusting develops when tailing surfaces dry out. In conventional spigotting and cycloning operations, large areas of tailing are left unwetted while the build-up of tailing sand is being carried out elsewhere. This permits complete surface drying with inherent dusting potential. Furthermore, with steep dams built of tailing, the water table is designed to drop fairly low in order to increase stability. Dropping the water table, however, allows quick drying of the dam slopes, resulting in dusting. In the thickened discharge method with a fixed discharge, the surface of the tailing area is wetted continuously. This virtually eliminates dusting. This process likewise reduces sublimation in the winter.

The absence of segregation may have a special advantage for uranium mining operations. Because radium tends to accumulate in the tailing 'slimes', environmentally undesirable concentrations of this ore are probable in a conventional tailing pond. Adopting the thickened tailing disposal method will ensure that the radium is dispersed uniformly throughout the mass of tailing, at a concentration that is even lower than in the original ore body.

REHABILITATION PROCEDURES

A comparison of the rehabilitation procedures for the conventional and thickened discharge disposal areas is warranted. Rehabilitation of a conventional disposal area involves:

- Considerable regrading of the entire perimeter dam slope.
- Construction of a toe filter to prevent erosion by seepage.
- Permanent plugging of the decant system.
- Construction of a surface drainage system to replace the decant.
- Preparation of the surface for vegetation by spreading and ploughing under: neutralizer, fertilizer and, where necessary, topsoil.

It is difficult and sometimes impossible to operate equipment on the flat, soft, poorly-drained surfaces of abandoned conventional tailing areas for many years after deposition. Yet, unless the surface is regraded to provide drainage, plant life cannot survive.

The problem does not arise in the rehabilitation of an area where the thickened discharge disposal system has been used. This system already provides a sloping terrain which is drained naturally and, therefore, does not require regrading. Shortly after termination of operations, equipment can operate on the surface. The overflow culvert for the tailing-water pond is located at the edge of the tailing deposit. It requires no major alteration for abandonment.

Rehabilitation of a thickened discharge disposal area makes use of the existing discharge system. Therefore rehabilitation is much simpler to achieve at a considerably lower cost, as outlined below:

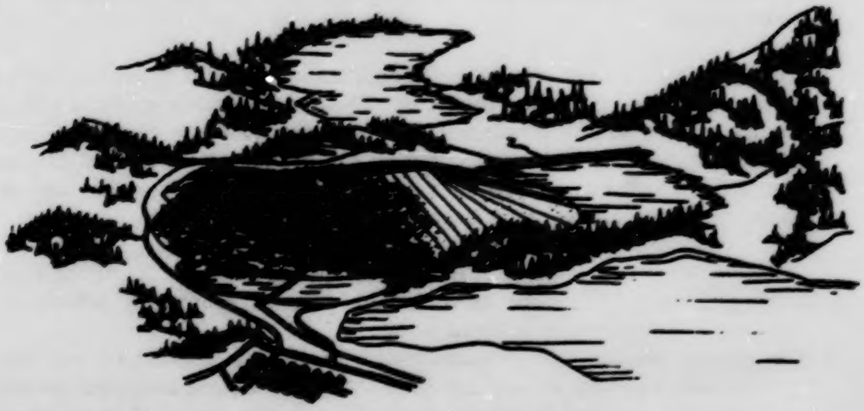
- Several months prior to abandonment of an area, appropriate neutralizer, fertilizer and possibly seed are added to the tailing being discharged to initiate growth on the area. Because tailing build-up occurs in parallel layers, (see next

Section), it is possible to develop a uniformly treated surface layer of preplanned thickness covering the entire surface. Thus a major portion of the rehabilitation process can be achieved automatically.

- When the deposit is developed longitudinally in a ridge, progressive treatment is possible and environmentally beneficial to 'close down' the completed portion of the deposit.
- On sloping ground or valley impoundment areas, it may be desirable to establish collector ditches to control run-off beyond the perimeter of the tailing deposit.
- The final steps in abandonment operations may require conventional contour plowing comparable to good farming techniques.

Figure No. 4 is a view of one of the proposed arrangements for tailing disposal at the South Bay division operation of SELCO Mining Corporation Limited in Ontario. The illustrated tailing pond will continue to be filled by conventional disposal methods for another year. Thereafter, the thickened tailing disposal method is proposed. The sketch illustrates the tailing deposit some years hence. Tailing discharge has been completed at the left end to create a 6 percent slope, and revegetation has been initiated. At the right end tailing disposal continues, progressively developing a long ridge of tailing. It should be noted that any fines that have been washed out by run-off are buried progressivley as the deposit slowly encroaches onto the pond to the right.

In connection with revegation of tailing disposal areas, advances are being made by the author's colleague, botanist Dr. T. C. Hutchinson of the University of Toronto. Instead of relying on continued neutralization and fertilization, Dr. Hutchinson has successfully sought out and is developing plant strains that thrive on the acidic environment of a tailing pond (4). The special plant strains, the excellent drainage and the low seepage characteristics of the thickened tailing slopes should reduce future rehabilitation problems to quite manageable proportions.



View of Tailing Disposal Area in the Future. SELCO Mining Corp. Ltd., South Bay Division.

Figure No. 4

DEPOSITIONAL BEHAVIOR OF THICKENED TAILING

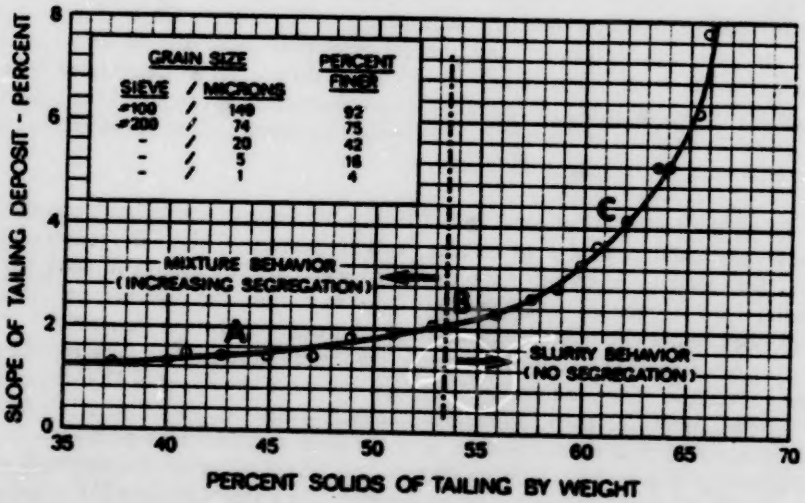
At the commonly used tailing concentration of 25 to 35 percent solids, segregation is rapid. The material at this low percent solids behaves like a mixture, each particle settling at its own rate, dependent primarily on its size and specific gravity, and the temperature and velocity of the transporting liquid. The coarser material settles out close to the discharge point. Progressively finer material settles out farther from the discharge, and most of the slimes (extremely fine particles) are invariably carried to the slimes pond. The slope of the major portion of the deposit is usually less than one percent and is generally concave in shape. This description of depositional behavior is typical for conventional tailing ponds. Particle segregation is used to advantage to supply coarser material at the perimeter of the pond with which to build the tailing dam. The flat depositional slopes are of little concern, and the creation of a slimes pond is accepted as an undesirable, but inevitable, feature.

On the other hand, the thickened tailing discharge method, which comprises a conical deposit of tailing without perimeter dams and without a slimes pond, is made possible by the natural depositional behaviour of thickened tailing. Generally, it would be anticipated that even with thickened tailing, the coarser fraction would settle out closer to the point of discharge in accordance with the laws of sedimentation. This segregation by size, however, does not occur because the tailing, after thickening, becomes, in fact, a heavy slurry. The resulting deposit is not unlike the sediments found naturally and referred to as 'wackes' or 'graywackes' by geologists (5). The tailing slurry flows until the angle of internal friction (viscosity) is able to overcome the gravitational and dynamic forces acting on the flowing slurry. The resultant slope is close to the natural angle of repose of the slurry. This angle is determined primarily by the internal properties of the slurry, such as grain size, its distribution, specific gravity and the concentration of solids. In addition, an extremely important variable is the velocity of flow which is determined by the quantity of discharge at any one point. The volume of flow can affect the build-up of slopes by as much as one-half to one percent. Temperature and pH appear to play a minor role, as yet poorly defined.

Of all the factors mentioned that influence the slope of the tailing deposit, the most practical to control is the percent solids in the slurry. This relates to thickening and the resultant behavior of the thickened material. The change in behavior of tailing from a 'mixture' at low percent solids to a 'slurry' at high percent solids is described below with reference to the illustrative example given in Figure No. 5. This Figure represents the results of the special laboratory deposition tests performed for the conversion of the conventional disposal system of the Superior Division of the Magma Copper Company to the thickened tailing method. Very little effect on the slope of the deposition is discerned until the percent solids reaches about 44 to 46 (Point A in Figure No. 5). From here on, the effect of further thickening is very noticeable. The material begins to behave more like a slurry and less like a mixture. The slope of the deposition increases at an increasing rate. At Point B, betweeen 53 and 54 percent solids, slurry behavior predominates. Slimes and sand settle out together. Clear water is found seeping very slowly from the entire surface of the deposit, being squeezed out under the action of consolidation of the deposited tailing due to its own weight. The slope is now fully controlled by the viscosity of the slurry. As the tailing is thickened further from Point B to Point C, the deposited slope increases extremely rapidly.

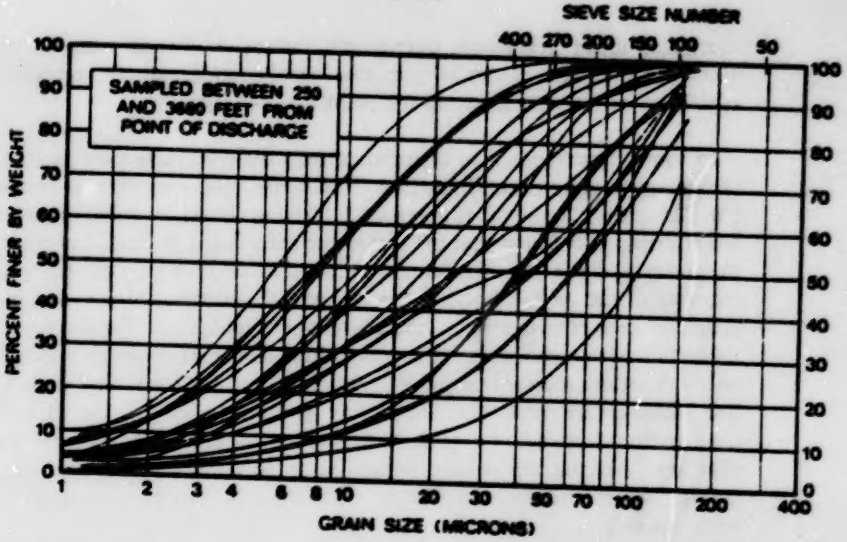
The ideal slope for a tailing deposit is believed to be about 6 percent. This will provide safe stability and excellent drainage for future plant growth. It has been found that there is a considerable variation in the degree of thickening required, from mine to mine, to achieve the suggested 6 percent slope. Early in the development of the thickened tailing discharge method, it was recommended that both laboratory and full-scale field tests be performed to establish the relationship between the degree of thickening and the developed slope (1). Experience and the development of special laboratory deposition tests now permit the establishment of this relationship without the need for field tests for most tailings. Generally, it has been found that 'slurry behavior' predominates at approximately 50 percent solids, and therefore the minimum thickening required will be near this value.

Studies confirm that when the tailing is thickened adequately, particle segregation does not occur in the field either. Grain-size analyses have been performed on tailing samples obtained from the disposal area of the Texasgulf Canada Limited Kidd Creek mine. Samples were obtained at 250-foot intervals from the point of discharge to a radial distance of 3680 feet, both before and after significant thickening had been commenced. Figure No. 6 illustrates considerable grain-size variation resulting from a tailing discharge at 37 percent solids. Figure No. 7 shows much reduced variation in grain size, the samples having been taken from the same sampling points, but after the tailing had been thickened to 50 percent solids. Figure No. 8

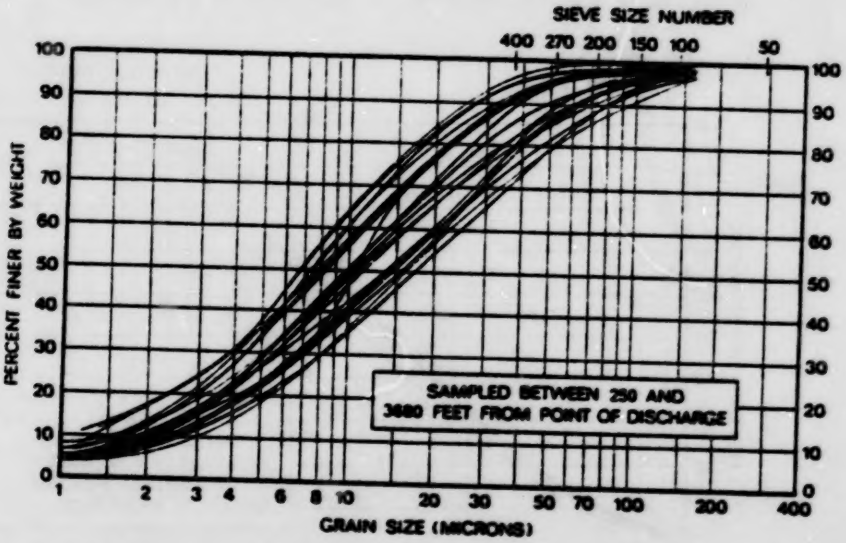


Laboratory Tailing Deposition Test Results. Magma Copper Company, Superior Division.

Figure No. 5



Grain-Size Curves of Tailing Discharged at 37 Percent Solids. Texasgulf Canada Ltd., Kidd Creek Mine.
Figure No. 6

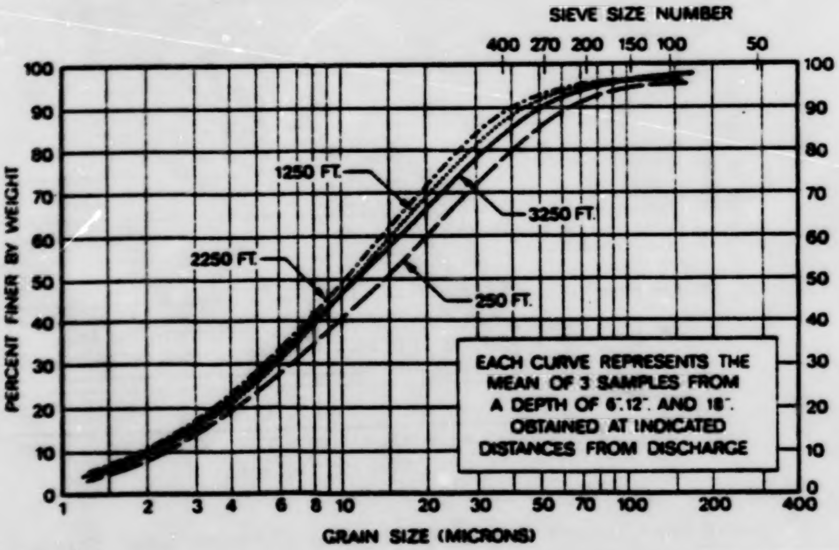


Grain-Size Curves of Tailing Discharged at 50 Percent Solids. Texasgulf Canada Ltd., Kidd Creek Mine.
Figure No. 7

illustrates mean grain size distribution curves of tailing samples obtained at 1000-foot intervals. It can be seen that hardly any grain-size segregation occurs, regardless of the distance from the point of discharge.

In some mines there is an operational requirement to use the coarser fraction of the tailing as mine-backfill material (commonly equivalent to that retained on the No. 325 screen). Removal of this coarser fraction results in a finer grain-sized tailing for disposal. It is found that the higher the percentage of fines, the more viscous is the slurry, and the less thickening is required to create a particular slope. Thus the removal of the coarse fraction has an overall beneficial effect. As thickening is generally an expensive process, the advantage is significant. Figure No. 9 illustrates the results of the special laboratory deposition tests performed on tailing, both with and without the coarser fraction, 'total tailing' and 'cyclone overflow' respectively. The samples are from the South Bay Division mine of the SELCO Mining Corporation Limited. To achieve deposition slopes of 6 percent the 'cyclone overflow' material will require thickening to 66 percent solids, whereas the 'total tailing', which includes the coarser fraction, will require thickening to over 69 percent solids.

The final element that makes the thickened tailing deposit a viable solution, is the absence of concavity on the surface of the tailing. Development of a concave surface would indicate the undesirable presence of segregation as well as a reduction in volume capacity for any particular discharge elevation. It is expected that the slope of the tailing must remain linear, as determined by the viscosity of the slurry.



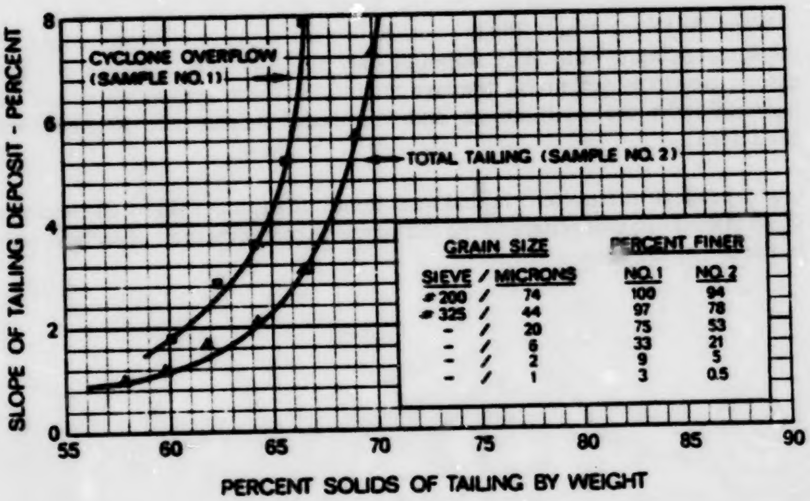
Grain-Size Curves of Tailing Discharged at 50 Percent Solids. Texasgulf Canada Ltd., Kidd Creek Mine. Figure No. 8

Because of the addition of water from the underlying tailing as it consolidates, the viscosity of the fresh tailing will decrease as the distance from the discharge increases. This change in viscosity, however, is small and does not affect the overall slope to any extent. A survey made of the tailing deposit at Kidd Creek Mine reveals a maximum concavity of the surface of only 2 to 3 feet over a maximum measured radial distance of 3680 feet.

In connection with the developing plane surface of the deposit at the mine, it should be added that an annual deposit is represented by a layer 2 to 4 feet in thickness over the entire tailing surface. Such uniformity is important for eventual revegetation purposes, discussed earlier.

INITIATING THE THICKENED DISCHARGE METHOD

Although the basic principles described are simple, it is cautioned that careful planning and design are required to achieve the advantages discussed herein. Not only must the location of the discharge points be selected carefully, but appropriate decisions must be made regarding the diversion of run-off water, and the degree of thickening that would be required. It must be emphasized that laboratory experimentation combined with field tests are the rational way of obtaining the necessary information for establishing the latter. The most costly item is the thickener, which must have the capacity to thicken tailing to above 50 percent solids. To reduce its size, and therefore its cost, to a minimum there is an advantage in examining the flow sheet in order to:



GRAIN SIZE			PERCENT FINER	
SIEVE	/	MICRONS	NO. 1	NO. 2
#200	/	74	100	94
#325	/	44	97	78
-	/	20	75	53
-	/	6	33	21
-	/	2	9	5
-	/	1	3	0.5

Laboratory Tailing Deposition Test Results. SELCO Mining Corp. Ltd., South Bay Division.
Figure No. 9

- Decrease all process water in the plant to a practical minimum.
- Convey all essentially clear effluents by separate pipeline to the tailing area and discharge them within the perimeter of the deposit. These effluents flow around the tailing cone and into the tailing-water pond for further treatment, if required.

As part of the planning and design program, it is also necessary to define the areas to be allocated for the tailing-water and reclaim-water ponds. Finally, all dams for these ponds must be designed properly using sections and materials that will assist in the eventual rehabilitation of the tailing area.

CONCLUSION

The thickened tailing discharge method, which develops a low-profile conical hill, provides financial and environmental benefits.

Achievement of these goals requires precise initial planning and design in order to choose the optimum disposal area; to determine the size and placement of the discharge points; and to establish the requirements for the thickening operation. In addition, natural run-off water interception must be appraised, the tailing-water pond sited and the appropriate dams designed. After implementation, periodic inspection is required to ensure program achievement.

When the tailing area is finally returned to nature, the cone blends naturally with the surrounding terrain. With slopes of only 5 to 8 percent, the deposit enhances natural growth by optimizing drainage conditions. Therefore, the watershed merges naturally with the environment and continued maintenance is minimized or eliminated.

Independent studies confirm that most mines stand to benefit by converting their conventional disposal system to the thickened discharge method as early as possible in the life of the mine.

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The value of a new technique in industry is best expressed by the confidence and enthusiasm with which it is accepted. The author wishes to express his sincere thanks to both management and staff of the following mining companies for permission to reproduce data pertaining to their operations.

Texasgulf Canada Limited, Kidd Creek Mine.

Consolidated Durham Mines and Resources Ltd.

Magma Copper Company, Superior Division.

SELCO Mining Corporation Limited, South Bay Division.

Discussion of Paper by E. Robinsky

N. Davé	What hydraulic conductivity do you expect?
E. Robinsky	The tailings will be 2/3 that of the slimes alone. If the sands are 1×10^{-3}, the slimes will be 1×10^{-5}.
N. Davé	What are the chances of erosion during spring thaw?
E. Robinsky	It seems to work well. Also the system appears to hide its mistakes. A 6% slope really is very gentle.
D. Francis	What happens at the end of the dam? Is there segregation?
E. Robinsky	No, there is no segregation. Think of it as a heavy cream.
C. Farrow	There is the question as to the stability on a year-round operation when the tailings are deposited in winter with the subsequent freeze-thaw. Also, with conventional un-thickened discharge there is segregation, so the discharge point is frequently moved. Is the spigot changed frequently in the thickened discharge method?
E. Robinsky	We have drilled and found layers of ice and there is segregation. During the flow the meandering stream vanishes as usual. The material is very uniform. There is no evidence of wash out due to melting. Kidd Creek Mines tails is 0.8% without thickening and 1.1 with thickening. Expected to behave similar to conventional discharge, and could go down to the perma frost. The topography will dictate the frequency of change of the discharge point and therefore the topography must be taken advantage of.
L. Melis	a) When I saw the Kidd Creek tails there was much evidence of erosion. b) Is there a possibility of using the method in Northern Saskatchewan, as 70% discharge is difficult to flow due to the high viscosity.
E. Robinsky	Regarding Kidd Creek, they are about 4 years behind schedule and are not thickening sufficiently and therefore eroding is occurring on the perimeter. They are now installing a second large thickener. Regarding Northern Saskatchewan and high viscosity, not a high percentage of solids would be required. In Quebec, 47% solids will be pumped.

Discussion of Paper by E. Robinsky (cont'd)

R. Swider	I understand Kidd Creek tested a centrifuge.
E. Robinsky	I am not aware of that.

DECOMMISSIONING OF THE BEAVERLODGE MINE-MILL OPERATIONS AND RECLAMATION OF THE SITE

BY

M. FILION
ELDORADO NUCLEAR LTD.

(NO PAPER AVAILABLE)

Discussion of Paper by M. Filion

N. Aspin	1) In the light of uncertain regulatory planning, how can you plan ahead? 2) If you had done nothing, how far from what was required would you now be?
M. Filion	1) As long as a reasonable and general criteria is provided, then a plan can be prepared. There must be options left open. 2) We would be reasonably close on many of the requirements.
L. Melis	How much was spent on pathways?
M. Filion	I can't say.
T. Edwards	What do you accomplish by demolishing the building?
M. Filion	It prevents the possibility for salvage of buildings, but we were told that this had to be done and the site to be handed over to the Province of Saskatchewan after 5 years of monitoring.

CLOSING DOWN OF
THE BEAVERLODGE MILL

by

D. Grant Feasby*

For presentation at Annual Meeting of Canadian Uranium Producers' Metallurgical Committee, Mississauga, Ontario, May 19, 1933

*Former Mill Superintendent

SUMMARY

After 29 years of operation, the Beaverlodge uranium extraction plant was closed in June 1982. The history of development, operation, and procedures employed to decommission the operation are briefly reviewed. Because the process had been based on an alkaline carbonate leach, special techniques had to be developed to clean out the plant and dispose of mill solutions.

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1.0 HISTORY OF OPERATION

The Beaverlodge metallurgical plant went in operation in April 1953 at a rated capacity of 450 tonnes per day, and it continued in operation until June 1982. During 29 years of operation, over 10 million tonnes of ore assaying 0.23% U_3O_8 (0.19% U) were processed. Average milling rate was 1080 tpd, although rates as high as 2000 tpd were achieved.

The property is located in the northwest corner of Saskatchewan (Figure 1) and access was by air from Edmonton and by barge from Ft McMurray. In later years, air transport originated in Saskatoon and a winter road was available from early February to mid-March.

The original process development, which required considerable pioneering of uranium metallurgy, had been undertaken jointly by the Radioactivity Division of the Mines Branch in Ottawa, a chemical extraction group at Eldorado's Port Hope refinery, and a research team at the University of British Columbia. The investigators included some outstanding contributors to uranium hydrometallurgy: Dr. C.S. Parsons, Mr. A.H. Ross, Prof. F.A. Forward, and Dr. A. Thunaes. Although an acid leach process had been developed for Port Radium, a pressure alkaline leach was chosen based on a projected operating cost of $7.69/ton for acid, versus $5.09/ton for alkaline. The higher projected costs for acid leach were primarily the result of high acid consumption and high costs for sulphur.

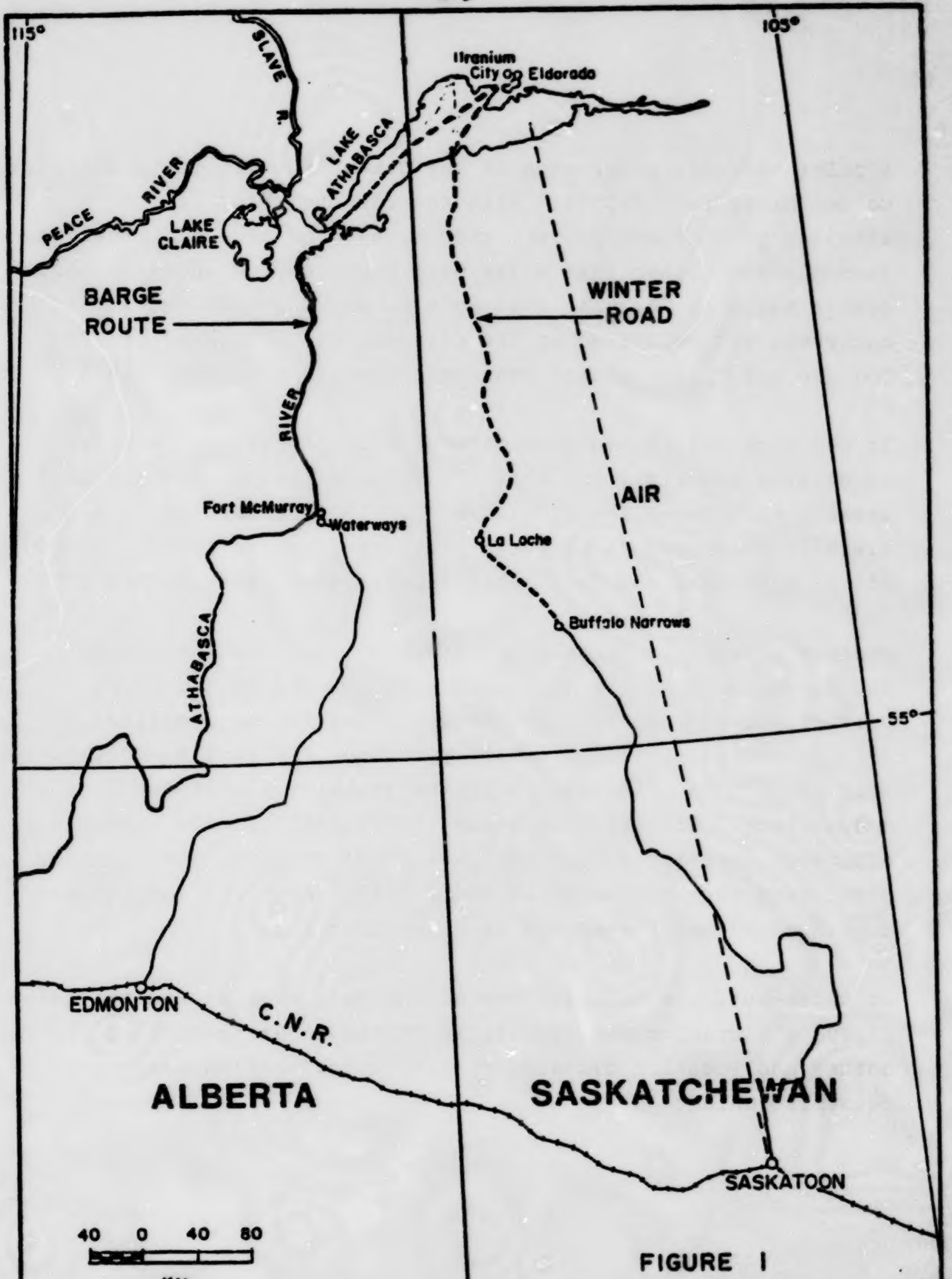
115°
105°
110°
55°
SLAVE R.
Uranium City
Eldorado
LAKE ATHABASCA
PEACE RIVER
LAKE CLAIRE
BARGE ROUTE
WINTER ROAD
RIVER
Fort McMurray
Waterways
La Loche
Buffalo Narrows
AIR
ATHABASCA
EDMONTON
C.N.R.
ALBERTA
SASKATCHEWAN
SASKATOON
40 0 40 80
KM

FIGURE 1
SITE LOCATION

A pilot test was undertaken at the Ottawa facilities of Sherritt Gordon Mines in mid-1951. Although the chemistry of the alkaline process was proven, the engineering of a flowsheet was incomplete. Rather than delay mill construction another year, design began in November 1951 by R.M. Way and Associates, and equipment was delivered to the site during the summer of 1952. The 450 tpd "pilot plant" went into operation in April 1953.

It was soon determined that corrosion of the 18 carbon-steel autoclaves was rapid and that the soluble loss of the two 18.3 m washing thickeners was excessive. In 1953, Eldorado's R & D division conducted atmospheric pressure leaching tests. Four 255 m^3 pachucas, and four extra filters were installed in 1954.

Process research continued in Ottawa on the alkaline system. The development of the high pyrite Verna orebody and other privately owned deposits in the area, led to the decision in 1955 to increase milling capacity to 1800 tpd with an ancillary acid process to acid leach a pyrite-graphite flotation concentrate. Construction began in the fall of 1955 with heavy equipment arriving during the 1956 barge season. The expanded plant went into operation in April 1957. However, the process continued to be changed and improved (Table 1).

At close-out, the mill was one of the most complex and extensive (2,500 m^2) uranium ore processing plants in existence. Its nature and location dictated special consideration for decommissioning.

TABLE 1

BEAVERLODGE MILL DEVELOPMENT

MONTH	YEAR	EVENT
May	1952	Construction begins
April	1953	450 tpd plant started
December	1954	4 pachucas, 4 extra filters, capacity to 650 tonnes
April	1957	Capacity to 1800 tpd with ancillary acid leach
January	1959	Lime plant started
February	1962	Ore sorting plant started
September	1964	No. 1 oxygen plant started
November	1964	Autoclaves shut down
January	1965	Autogenous mill started
September	1968	Ore sorting plant down
September-December	1970	Pachucas to mechanical agitation
February	1973	Auto mill down because of cracks
September	1975	New auto mill installed
November	1975	No. 2 oxygen plant started
	1978	Major upgrading of structures and systems started
March	1979	Run-of-mine sample plant
May	1979	New carbonation plant
November	1979	New heat exchange
September	1981	New vacuum system

2.0 MILL OPERATIONS, DECEMBER 1981 TO JUNE 1982

2.1 GENERAL

On December 3, 1981 it was announced that the operation would close on 30 June 1982, a lead time of 7 months.

At the time of the announcement, ore supply was poor and throughput was being supplemented by processing ore sorter rejects. Rehabilitation of the plant was continuing with filter overhaul and the installation of new flotation cells.

The following actions were taken immediately:

- contract overhaul stopped
- mill training stopped
- inventory of supplies taken
- budget and shutdown plans drawn up
- autogenous mill used primarily
- circuit modified to reduce solution volume
- test work started on solution neutralization

2.2 OPERATING HIGHLIGHTS

2.2.1 Ore Supply

Grade and tonnage of ore during shutdown was substantially above expectation. Tonnage processed in March was the highest in 13 years. Yellowcake production was 36% above target. At the end of June 3,900 tonnes of ore remained in bins.

2.2.2 Grinding

Autogenous mill gave the best performance since installation. Grinding-thickening capacity was shown to be double three years earlier. Steel addition to mills was stopped in mid-May.

2.2.3 Recovery

Metallurgical recovery dropped from over 90% to less than 80%. Although plant leaching conditions were weakened, laboratory tests (alkaline and acid) confirmed poor extraction was due to brannerite-rich ores from the lower levels of the (Fay) mine.

2.2.4 Acid Circuit

The sulphide flotation and acid-leaching circuit was shut down in April to reallocate manpower, conserve acid, and increase sodium carbonate consumption.

2.2.5 Manpower

Operating personnel decreased from 60 to 35, requiring introduction of a 5-day schedule in May. There were no lost time accidents or union grievances.

3.0 DECOMMISSIONING

3.1 THE TASKS

The accepted definition of decommissioning was the implementation of measures to render the mill facility permanently out of service. These tasks were:

- removal of ore solids and leached solids from the plant;
- extract uranium from mill solution as yellowcake;
- remove residual yellowcake from vessels and equipment;
- develop and implement procedures to dispose of mill solution;
- remove all saleable supplies, equipment and structures; and
- develop and implement procedures for abandoning or dismantling the building, and reclaiming the site for the long term.

3.2 PROJECT TIMING AND PLANNING

From an operations point of view, it would have been preferable to complete the plant decommissioning in the shortest possible time with a full operations and mechanical crew. However, because of logistics of community support, it was decided that the work should be spread over a 12-month period starting June 1982. Fifteen man-years of work were identified.

Early in the planning, the tasks of keeping the mill and the tailings line warm were identified as major concerns. The maximum heat load was identified to be:

	MW	10^6 BTU/hr
Mill heat	1.69	5.78
Mill ventilation	1.38	4.72
Tailings line	2.65	9.06
TOTALS	5.72	19.56

The projected energy requirement was well within the capacity of the oil-fired boilers (16.5 MW) and an electric boiler (5 MW), but there was considerable unease about the vulnerability of the plans presented by the 3800 M Sclair tailings line. The line was rigidly held to a wooden trestle and cooling could result in collapse. Accordingly, contingency plans were drawn up to have the major effluent-producing activities completed by November 25, at which time the tailings line would be disconnected.

Fortunately, permission was obtained to commence allowing the mine to flood. After ore solids had been removed from the plant, tailings pumping could cease, and miscellaneous effluents could be redirected to the mine.

The planned schedule encountered considerable slippage at the beginning of the decommissioning phase because of the 3900 tonnes of ore left in the bins.

Decommissioning manpower for the mill was:

Supervision	5
Mill Operators	7
Mechanical	6
plus up to.......	6 temporaries

A chemist was retained for four months to monitor mill solution chemistry and to account for yellowcake production. A reagent operator was also retained for four months to prepare caustic solution for the precipitation circuit.

3.3 ORE SOLIDS REMOVAL

At shut down, the mill contained:

Ore in bins	3,900
Leached solids in pachucas	4,400
Thickener beds	2,300
Miscellaneous sumps	200
TOTAL	10,800

3.3.1 ORE BINS

There are two separate ore bins (Figure 2) in the Beaverlodge mill. A fine-ore bin of 2600-tonne crushed ore capacity was the feed surge for the primary ball mill circuit. A coarse ore bin of 5000-tonne capacity was the feed surge for the autogenous mill. Dead load in the bins was typically 450 t and 1200 t, respectively.

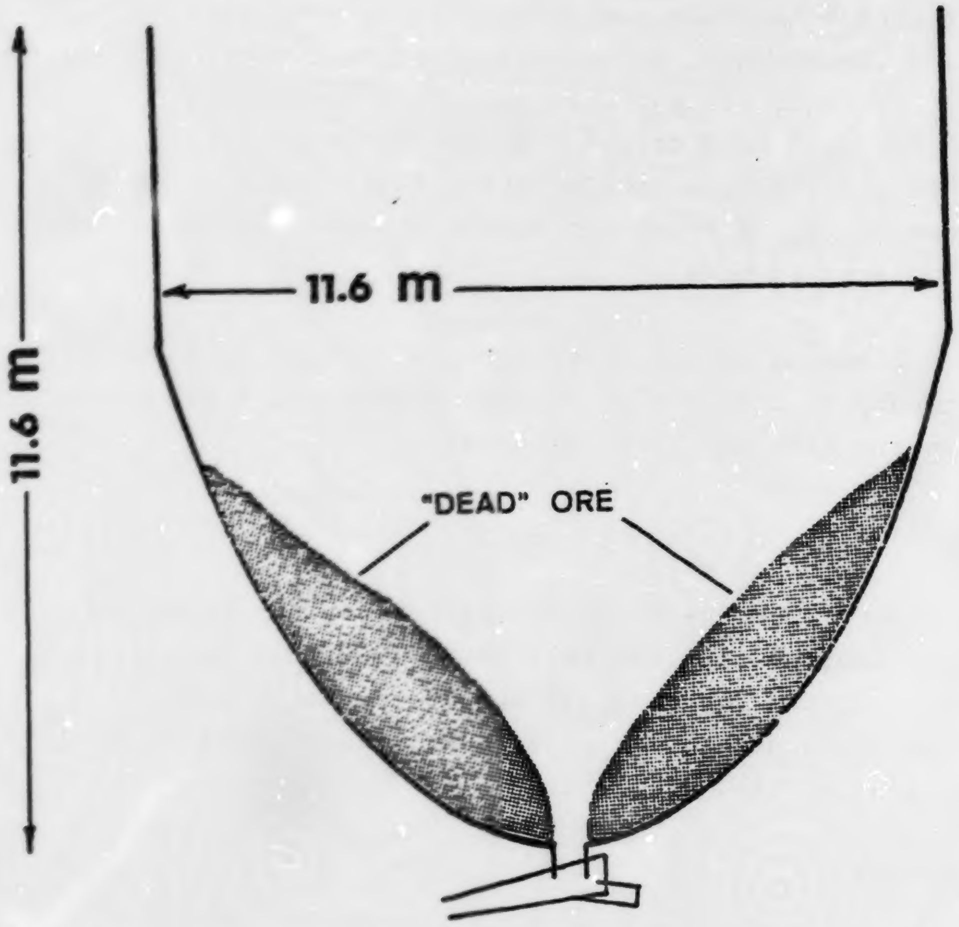
11.6 m
11.6 m
"DEAD" ORE

FIGURE 2
CATERNARY ORE BIN
BEAVERLODGE

It was decided that the ore should be ground out and processed for uranium recovery. However, operation of the plant with only 10% of normal manpower complement led to a plugged up thickener and blocked heat exchangers after four days. The standby thickener was put into service and grinding continued. The dead load of the bins proved to be very stubborn. Explosives were ineffective. Water hoses resulted in buried conveyors. A 5-ton air-driven slusher was installed and slow progress was made.

However, bin cleaning ceased after 28 days of very hard work when the standby thickener also failed. About 600 t had to be abandoned in the bins for later disposal.

3.3.2 Pachucas

Removal of slurry from the pachucas was routine and performed on schedule. Emptied pachucas were backfilled with grinding solution and agitated for residual solids removal. After draining, the 27 pachucas were water-washed and filled with barren solution awaiting disposal.

3.3.3 Thickeners

In the order of 2300 tonnes of solids represented a large amount of cleanup labour for the 8 trays. Following the conversion of the mill solution to barren, the mill grinding solution circuit was converted to water and conical fire nozzles connected to a 650-kPa supply which were used to undercut the 0.5 M beds. Working on a 3-shift basis, the washing of the beds was completed in 16 days.

3.3.4 Miscellaneous Leached Ore Solid

Using a 7-1/2 HP propane powered Bobcat and high pressure water, sump solids were also washed to tails. A general washing of the plant was undertaken and the ball mills were turned for several hours to decontaminate the liner and ball charge.

During the 29-year operation phase, cleanup was routinely attempted by washing with mill solution. Throughout the plant, deposits and stalactites formed which were only sparingly soluble in the solution. These were identified as:

Trona	$NaHCO_3.Na_2CO_3.2H_2O$
Burkeite	$2Na_2SO_4\ Na_2CO_3$

Both salts are readily soluble in water, and plant washing was efficient in removing surface deposits. However, repeated washing of concrete was ineffective in removing the entrained salts.

3.4 YELLOWCAKE SCALE

The precipitation circuit was unusually large for a uranium mill, with 900 m^3 volume in 12 vessels. The normal dilute yellowcake slurry was processed through the presses and loose sediments washed out with water. However, there remained on the inside of the tanks a very stubborn scale which resisted the following methods of removal:

- sledge hammers (outside)
- chipping hammers (inside)
- high pressure water (47 MPa, 7000 psi)
- explosives suspended inside.

A novel discaling technique was tried and proven effective. A detonating cord, called "B-line"* was wrapped around the vessels at 0.5 m intervals and exploded with 2 millisecond delay between wraps. Approximately 13,500 kg U was recovered from the vessels.

3.5 MILL SOLUTION

The most unique and challenging feature of decommissioning an alkaline leach plant is the disposal of large quantities of soluble salts contained in the recycled mill solutions. In normal operation, the Beaverlodge mill would contain:

Volume - 10^4 m^3 (2.2 x 10^6 gallons)

Na_2CO_3	550 t
$NaHCO_3$	160 t
Na_2SO_4	600 t
NaCl	45 t
U_3O_8	25 t
Ra^{226}	300-500 ug
pH	9.5 to 13

---

*Marketed by CXA Ltd., Tappen, B.C.

Of the dissolved salts, only uranium and radium could be substantially removed from solution.

Disposal of the mill solution had to be determined in advance of shutdown in order to prepare processing routes. The options for disposal were:

- evaporation to dryness and storage in containers underground:
  This option was not studied in detail but it appeared that evaporation would be very costly (~ $1 million) and that the mine itself could be a suitable container at depth;

- discharge of solution to tailings:
  During normal operation, between 20 and 50 m^3/day of mill solution (equivalent) was lost to tailings as "soluble loss". At the lower rate of 20 m^3/day, the salt loading to the environment would be equivalent to 500 operating days. Given that the tailings lake had a volume of 8×10^6 m^3, sodium levels would rise 80 mg/L above the normal operating level of 400 mg/L. Although uranium and radium loading would be less than cumulative of a year's operation, a 20% increase in salt levels in the tailings lake was inconsistent with the stated objective of minimizing environmental effects;

- disposal of solution to the mine:
  Following the decision (and permission) to permit the mine openings to flood, disposal of the solution in the lower levels of the mine was proposed to, and accepted by the regulatory agencies. Action taken was:

  - reduced solution volume from 10,000 m^3 to 6,500 m^3 during the operating phase;
    processed all of the mill solutions through the precipitation circuit to maximize uranium recovery and stored in the pachucas as alkaline barren; when salvage completed underground;
    pumped solution to the mine at the rate of 500 m^3/day.

Consideration had been given to the conversion of the contained carbonate salts to sulphate and bisulphate by the addition of sulphuric acid (some acid left over from acid circuit). However, this was rejected as exposing workers to unnecessary risk and providing little or no environmental improvement.

4.0 SALVAGE

Salvage of material and equipment for sale from the mill has been hampered by:

- old or specialized items
- poor market conditions
- contamination
- location

However, new company activities at Blind River, Ontario and Rabbit Lake, Saskatchewan have permitted relocation of items such as laboratory equipment and pumps. But it is difficult to justify the cost of removal and decontamination of a 25-year old ball mill, even though it may be in excellent condition.

Equipment removed from the property for sale has been subjected to the following criteria:

visual - no surface contamination
α , β < 3.7 kBq/m^2
- 10^{-5} uCi/cm^2
γ = twice background

Decontamination has been achieved by steam and high pressure water (47 mPa) cleaning. Some difficulty has been experienced with decontamination of mill equipment that has been exposed to mill solutions for several years.

5.0 DECISION ON STRUCTURES

At time of writing, no decision had been made on the ultimate disposition of the building. Ultimately, it is probable that the structures will be razed and the site landscaped. Some factors being considered are:

- large and complex structures
- massive concrete pads and foundation
- 31 very large steel vessels
- no wood
- contamination of floors and subfloors
- location on edge of major watershed
- safety of abandoned building
- poor future potential of contained equipment.

ACKNOWLEDGEMENT

The author wishes to thank Eldorado Resources Limited for permission to present the information contained in this paper.

DGF:fh
83.05.10

Discussion of Paper by G. Feasby

C. Farrow	Are all the diluted solutions put down the mine ?
G. Feasby	Yes

Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

CANMET
Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

ACTIVITIES OF THE NEA/IAEA WORKING GROUP ON URANIUM

M.C. Campbell
Extractive Metallurgy Laboratory

MAY 1983

For Presentation at the CUPMC, May 19-29, 1983
and for Publication in the Proceedings

Project ERP-4.4.1.0.01
Uranium - Conventional

ENERGY RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES
DIVISION REPORT ERP/MSL 83-72(OP&J) DRAFT

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ACTIVITIES OF THE NEA/IAEA WORKING GROUP ON URANIUM EXTRACTION

Although the past year has featured a considerable downward turn in uranium exploration, production, prices and future planning, there remains considerable international interest in uranium production. Perhaps uranium resource discoveries present a dilemna: to develop or not to develop. In this environment, the joint NEA/IAEA Working Group on Uranium Extraction (WGUE) has been quite active.

GREEN BOOK

In April 1983, the Nuclear Energy Agency of Paris announced the publication of "Uranium Extraction Technology". This book was initiated by the WGUE in 1978 under the Chairmanship of Mr. Henry James of South Africa. Contributions were received through assignments to various countries and were later edited by Dr. Derek Taylor of the NEA and Mr. Don Seidel of the IAEA in conjunction with representatives of the participating countries. Many members of the Canadian Uranium Producers Metallurgical Committee provided data, contributions to the text and editing of contents. It is generally an excellent book. Some sections are not as strong as they could have been because of concerns about sensitivities if the texts were changed radically; nevertheless, it is very comprehensive and informative about uranium processing in the Western World.

The list of contents is attached as Appendix 1. The chapters are:

I. Uranium Resources: Geology and Exploitation of Deposits. This provides brief descriptions of the major geological types of uranium deposits and the general metallurgical approach used for uranium extraction from them. It relates similarities and differences.

II. Advances in Technology: A sequence of advances in ore treatment technology is presented from feed preparation to tailings disposal. This is a review of recent developments rather than a comprehensive text on all unit operations.

III. Economic Assessment: This is an outline of the basic elements of cost evaluation. It provides a sequential logic for evaluation, an indication of the data requirements and the accuracy and sensitivity of the methodology.

IV. National Reports: These are descriptions of many of the uranium plants operating throughout the world. Summaries provide location, geology, mineralogy, mining and metallurgical practice, waste treatment and capacity data.

This is a valuable addition to your technical library and at $20 U.S., it is a bargain. It is available in Canada from the OECD agent:

Renouf Publishing Co. Ltd.
2182 St. Catherine St. W.
Montreal, P.Q. H3H 1M7
Telephone (514) 937-3519

WORKSHOP ON ECONOMICS OF URANIUM ORE PROCESSING OPERATIONS

This workshop was held in Paris on April 25 and 26, 1983 with a field trip to the COGEMA/SIMO uranium operation at Lodeve, France on April 27, 1983. As both Chairman and the lone Canadian at the workshop, I felt a rather heavy responsibility. Invitations were extended to many Canadians via the CUPMC list but the high cost could not be justified by other Canadians.

Forty people, representing 13 countries, who were in attendance at the workshop were welcomed by Mr. Howard Shapar, an American who is Director-General of the Nuclear Energy Agency. Mr. Shapar noted that in the 25th anniversary year of the NEA, the agency continued to encourage information flows between the countries of the OECD. He encouraged the workshop as another facet in demonstrating the economic advantages of the nuclear cycle.

Thirteen papers were presented in English or French. The titles and authors are shown in Appendix 2. The NEA will produce proceedings which will be available in about four months with the papers in the language in which they were presented. Proceedings will be available from the OECD at a nominal cost.

The papers fell into two general types:

(i) costing techniques, philosophies and methodologies, and
(ii) specific cases of costing evaluations.

Mr. Simonsen from Mintek in South Africa presented a general introduction to the practice of engineering economics. Mr. Coleman of Hazen Research outlined the factors for estimating uranium plant costs. Mr. Floter of Uranerzbergbau of Germany described the cost control system used for the Key Lake project. Mr. Moral of Spain and Mr. Fiuza of Portugal presented methods for assessment of new mine prospects for cut-off grade determination, mine profitability and preliminary processing costs. Messrs. Campbell of Canada and Devergie of USDOE (Grand Junction) presented reviews of application of cost studies in analysing domestic resources. Cost analyses with data summaries were presented by: Gomez of Spain on the ENUSA heap leaching operation; Hirono of Japan on vat leaching of a low-grade uranium ore; Roullier of Pechiney (France) on evaluation of innovative options (peroxide precipitation and alkaline leach); Hebert described the Lodeve plant costs; and Jug gave costs on upgrading the Gabon plant of COMUF.

It was a mixed group of papers but provided considerable insight into the use of costing methods. Data from various countries and operations provide useful bases for comparison. The proceedings should be a valuable reference document for anyone involved in process evaluation. I will notify CUPMC members whens the proceedings are available.

LODEVE PLANT VISIT

This plant is located in the south of France, in beautiful wine growing country, about 50 km from the Mediterranean Coast. The plant was well described in Mining Engineering (vol 144, No. 2, Feb. 1981), so I will not go into detail.

However, the plant is unique. The flowsheet is illustrated in Fig. 1. There is 2-stage alkaline leaching. Belt filters are used throughout. In order to meet environmental regulations, all sulphate in the system is crystallized out as commercial sodium sulphate. To meet purity specifications, molybdenum is recovered from the IX barren liquor and all organics are burned off. Tailings are sent to the abandoned open pit as "dry" filter cake by a conveyor belt. This plant is especially interesting because it has required a great deal of innovation.

SIXTH MEETING OF THE WORKING GROUP ON URANIUM EXTRACTION

This meeting was held at the OECD in Paris on April 28 and 18, 1983. Eleven countries plus the Nuclear Energy Agency (OECD) and the International Atomic Energy Agency (UN) were represented. The details of the meeting will be available about mid-summer in the NEA Newsletter "R & D in Uranium Extraction Technology", to which I hope you have subscribed or will do so at no charge.

Highlights of the meeting were as follows:

(a) Newsletter: The first newsletter of the NEA on R & D in Uranium Extraction Technology was issued in December 1982. It is available in English or French to anyone who requests it from:

The Secretary
Uranium Extraction Technology
OECD Nuclear Energy Agency
38, boulevard Suchet
F-75016 Paris, France

It comprises reports on R & D activities in countries participating in the working group, an article on the Kvansfjeld uranium ore of Greenland, on other activities in the NEA, and on recent publications and forthcoming meetings. This author has a few spare copies which include an order form.

(b) Research Reports: These will be reported upon fully in the Newsletter No. 2.

(i) South Africa: Mr. Simonsen of Mintek reported on recent experiences which demonstrated that a column leaching test method gave more consistent and meaningful results than the conventional rolling bottle method. Resin-in-pulp work at Ergo pilot plant has been beset with problems of resin breakdown which are being resolved using an air lift and a flooded screen separation technique. Mintek has developed an inventory of 10 reference uranium bearing materials and are working on an eleventh specifically as a radiometric ore sorting standard.

(ii) Australia: Dr. Levins reported on developments in Australia. Most R & D is confidential to the companies, but CSIRO does some work. CSIRO has worked closely with Interox on Caro's acid use at Narbarlek where a plant is now being built using Caro's acid. A major effort is in radium removal so a solubility increase from 300 p Ci/L to 1500 p Ci/L is a significant element of this work.

(iii) Denmark: Dr. Jensen reported on work at the Danish National Laboratory on the Greenland ore. About $10 million U.S. has already been spent, much of this in the installation and operation of a Luigi pipe reactor. This reactor is about 2 km long and 25 mm in diameter. Pressure is applied by the feed pump. Discharge is through a 10 mm throttle at the end.

The pipe reactor was installed in December 1981 with a capacity of 2.2 tonnes per hour. It has six zones in which the temperature is controlled. Maximum conditions for the carbonate leach are 290°C and 20 atmospheres. Recoveries of 86-92% are achieved with the 20 minute residence time. A teflon lining has been installed to prevent scaling. The balance of the circuit is comprised of a belt filter, ion exchange and caustic precipitation.

The National Laboratory is seeking a simple circuit for Greenland applications. As yet there is no private sector interest in this Greenland ore, a prerequisite to mine development.

(iv) Spain: Dr. Moral reported that Dr. J. Josa of the Junta Energia Nuclear in Madrid had accepted the IAEA post formerly held by Mr. Don Seidel (who has returned to the USBM in Salt Lake City).

Dr. Moral reported research on bacterial oxidation to produce ferric iron for leaching. Tests showed that 2.5 kg Fe^{3+}, 12 kg H_2SO_4 at pH 2 yielded 92% uranium recovery in 14 hours. Work continues on recovering 400 ppm of uranium from lignites, but little progress has been made. The JEN has completed work on the purification of a Mexican U_3O_8 containing Mo, V and SiO_2. JEN has developed a mathematical model for a plant for uranium recovery from phosphoric acid including mass and energy balances.

(v) U.S.A.: Mr Paul Devergie of the USDOE Grand Junction, Colorado reported briefly on the depressed state of the U.S. industry. He did not feel he was qualified or prepared to speak on R & D but would attempt to report for the Newsletter .

(vi) France: Mr George Lyaudet of Cogema reported on R & D in France. A continuing study of phosphoric acid circuits is showing good results with a new solvent to replace DE2HPA with 3 times the loading capacity. A new deposit in the Bordeaux region presents a challenge because the uranium is in a clay matrix which makes liquid/solids separation extremely difficult. CCD looks like a possibility and resin-in-pulp is being evaluated. A pilot-plant study is underway on a uranium-bearing molybdenum ore to produce uranium plus a high purity molybdenum product.

(vii) Algeria: Messrs Kebir and Kaddour, both recent graduates from MIT, reported on a deposit in the Hogar region of Algeria. There is no water within 350 km of the deposit, the region has had no rain in 5 years. It contains 0.1-0.8% U and similar amounts of Mo in a silicate matrix with some carbonates although the mineralogy is largely undetermined. Acid leaching tests indicate 350 kg H_2SO_4. Pressure alkaline leaching tests at 230°C with 100 mesh ore gives good dissolution. Liquid/solids separation with limited water presents difficulty. Alternative systems such as preconcentration and shipment to a less arid area will be studied.

(viii) Mexico: Mr. Patino Guerrero reported on the fledgling Mexican uranium industry. Six areas are being studied with two sites being prominent: Pena Blanca in Chihuahua, and at the Bulgas Basin. The Pena Blanca operation would involve 2 open pit mines and one underground. Processing would be conventional acid leach with CCD followed by SX, carbonate strip and caustic precipitation. Investment would be $62 million and operating costs $20 per pound of U_3O_8. Startup, which was to be August 1982, is now 1985.

The Bulgas Basin deposit presents a more difficult situation. Alkaline leach of the 800 g/tonne ore is indicated. Liquid/solids separation difficulties may require radical treatment including roasting at 380°C followed by quenching. It would be used for uranium recovery.

Another deposit could be amenable to heap leaching. A phosphoric acid plant on the Baja peninsula may be a further development.

(ix) United Kingdom: Dr. Triggs reported that current R & D at Warren Spring Laboratory on recovery of U from phosphoric acid will probably end in late 1983 when funding by the EEC runs out.

(c) Related Activities:

(i) "Advances in Uranium Ore Processing and Recovery from Non-Conventional Resources"
Technical Committee Meeting
Vienna, Austria
26-39 September 1983

Information sheets are available from this author. This meeting has had a poor response from North America. Some 19 papers have been confirmed to date and 50 people are registered. However, there are no Canadian papers or delegates.

Papers received to date cover: heap leaching, the pipe reactor. Key Lake (Uranert) sea water, phosphoric acid, L/S separation, SX & IX and percolation leaching. Additional papers and registrants are being encouraged. There is no registration fee.

(ii) "International Meeting on Recovery of Uranium from Seawater"
Tokyo, Japan
17-18 October, 1983.

This is an update in R & D since the MIT meeting in December 1980. It is sponsored by the IAEA and the Atomic Energy Society of Japan.

(iii) Environmental Working Group:

Mr. Peter Rafferty of the NEA reported that the group of experts on tailings has nearly completed its work. The work has been considered a success in that it has demonstrated that the ICRP standards cannot be directly applied but provide useful guidelines. There has been no final technical answer to tailings disposal, but the forthcoming NEA report will be a significant contribution to the "state-of-the-art".

Tailings facilities do require special attention particularly to erosion by water. Siting must be in a geomorphologically stable area. The retention facility design is very important. Good drainage and a stable cover could help avert water erosion.

Mr. Rafferty's term at the NEA is now complete, and he is seeking employment in the U.S.A.

(d) Future Activities of the NEA/IAEA Working Group on Uranium Extraction.

(i) Location of Meetings: In the future attempts will be made to locate the Working Group meetings in areas other than Vienna and Paris. The lack of participation by industry in the Working Group could be altered by having meetings in more convenient locations where local industrial representatives could attend. Canada would be an appropriate location perhaps in conjunction with the 1984 or 1985 CUPMC meeting, or with the CIM Conference of Metallurgists. Other locations might be the U.S., Australia and Spain. A decision on the venue for the 1984 Working Group meeting will be made by September 1983.

(ii) Index of International Uranium Extraction Costs: Mr. Simonsen of South Africa proposed that a "shopping basket" of items might be dentified which could provide an index of comparative costs internationally. Messrs Simonsen and Devergie (USDOE) will collaborate on assembling and testing a list of items.

(iii) File of Production Data: In 1980, Messrs Hartley (IAEA) and James (South Africa) assembled a compilation of production data on operations around the world. These data were used as the basic information for the IAEA Technical Report Series No. 196 entitled "Significance of Mineralogy in the Development of Flowsheets for Processing Uranium Ores" and for the "Green Book" described earlier in this report.

There is now interest in updating this file of data with initiation at the 1984 Working Group meeting.

(iv) Manual on Laboratory Evaluation Techniques for Uranium Ore Processing: This is being prepared by IAEA and is well on its way with a planning group calling on individuals to contribute. It is intended to be a guide to those relatively new to uranium ore evaluation, but may be useful to others when available later this year.

(v) Manual on Cost Evaluation: It is intended that the IAEA will utilize the experience and information of the Workshop held at the NEA on April 25 and 26, 1983 as the basis for preparation of a manual on cost evaluation of uranium deposits. A group of experts will be assembled to put together a relatively easy to use methodology as a guide in assessing deposits. This would be of use to geologists, mining and extractive personnel especially in new situations.

(vi) Next Working Group Meeting: It has been recommended that that next meeting be held in one of the member countries rather than at the IAEA in Vienna or the NEA in Paris. Volunteer hosts are being sought.

A workshop on productivity in the uranium industry has been proposed with general topics such as:

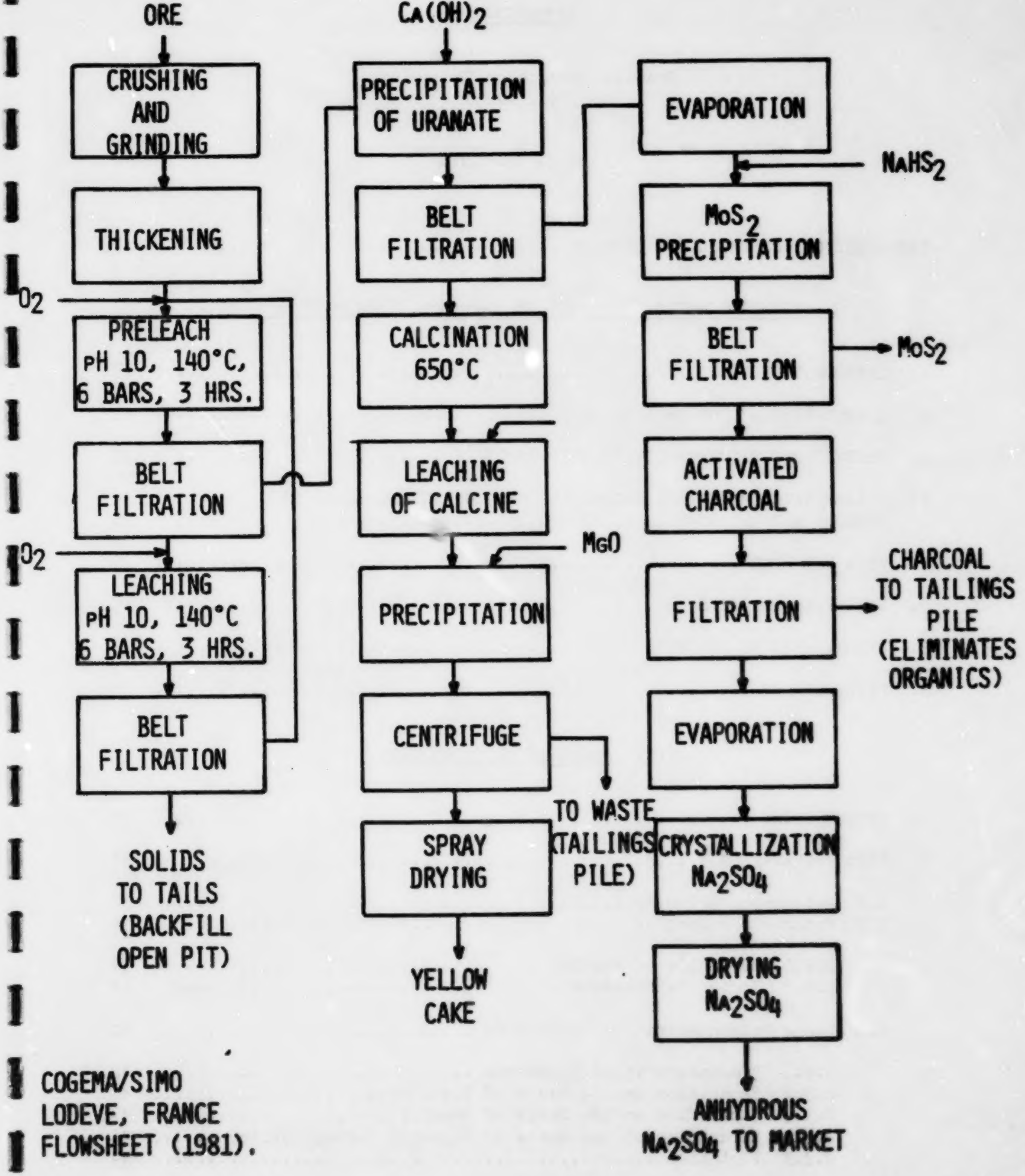
- grade control
- instrumentation and process control
- maintenance and maintenance strategies
- training for productivity

The theme would be geared toward survival or coping with the economic downturn.

COMMENT

Canada is looked upon as a leader in uranium production and technology. It is unfortunate that there is little Canadian participation in this international forum. The workshops and meetings, and the reports on them in the Newsletter provide international exposure. Such exposure can provide international contacts for both promoting and acquiring new technology. A meeting of the Working Group in Canada in conjunction with a technical meeting on uranium could be a powerful means of enhancing the Canadian image.

FIGURE 1


ORE
CRUSHING AND GRINDING
THICKENING
O2
PRELEACH pH 10, 140°C, 6 BARS, 3 HRS.
BELT FILTRATION
O2
LEACHING pH 10, 140°C 6 BARS, 3 HRS.
BELT FILTRATION
SOLIDS TO TAILS (BACKFILL OPEN PIT)
CA(OH)2
PRECIPITATION OF URANATE
BELT FILTRATION
CALCINATION 650°C
LEACHING OF CALCINE
MGO
PRECIPITATION
CENTRIFUGE
TO WASTE (TAILINGS PILE)
SPRAY DRYING
YELLOW CAKE
EVAPORATION
NAHS2
MoS2 PRECIPITATION
BELT FILTRATION
MoS2
ACTIVATED CHARCOAL
FILTRATION
CHARCOAL TO TAILINGS PILE (ELIMINATES ORGANICS)
EVAPORATION
CRYSTALLIZATION NA2SO4
DRYING NA2SO4
ANHYDROUS NA2SO4 TO MARKET

COGEMA/SIMO
LODEVE, FRANCE
FLOWSHEET (1981).

APPENDIX 1

Uranium Extraction Technology
(OECD 1983)

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APPENDIX 2

WORKSHOP IN ECONOMICS OF URANIUM ORE PROCESSING OPERATIONS

NEA, Paris, France
April 25-26, 1983

H. Shapar, NEA; Welcoming Address.

1. H. Simonsen, Mintek, South Africa;
   "The engineering economics of uranium processing".

2. J.P. Hebert, Cogema, France;
   "Etude des couts d'investissement pour l'usine de traitement de minerai COGEMA de LODEVE".

3. M.C. Campbell, CANMET, Canada;
   "Uranium extraction casting in Canada".

4. J.P. Roullier, Pechiney-Ugine-Kuhlmann, France;
   "Estimation des effects de l'innovation sur les couts par un snodele technica economique d'usine de concentration de nimerai d'uranium".

5. P. Devergie, USDOE;
   "Overview of DOE cost estimates on uranium production".

6. J. Moral, JEN, Spain;
   "Cost and sensitivity exercises with a pocket calculator".

7. R. Coleman, Hazen Research, U.S.A.;
   "Unit factors for cost estimation in uranium ore processing".

8. A.M.A. Fiuza, Empresa Nacional de Uranie, Portugal;
   "Preliminary optimization of grades for alternate and/or complementary exploitation and processing methods".

9. P. Gomez-Juan, ENUSA, Spain;
   "Cost methodology for open-pit and heap leaching operation".

10. S. Hirono, Power Reactor and Nuclear Fuel Development Corp., (PNC), Japan;
"Vat leaching of the uranium ore from Ningyo-Toge deposits".

11. A.M.A. Fiuza, Portugal;
"Determination of leaching recovery functions in a deposit with different regional behaviour (a case study of the Nisa deposit)".

12. F. Floter, Uranerzbergbau, Germany;
"The unique process for the Key Lake Project, Canada".

13. V. Jug, Cogema, France:
"Building of a new plant by the COMUF in Gabon".

Discussion of Paper by M. Campbell

G. Feasby	What is the grade of the Greenland ore?
M. Campbell	500 ppm U_3O_8.

REDUCTION OF TAILINGS RADIUM LEVELS USING MODIFIED MILLING PROCESSES

BY

M.H.I. BAIRD
McMaster University

UPDATED REPORT ON N.S.E.R.C STRATEGIC GRANT TO MCMASTER UNIVERSITY

Presented by M.H.I. Baird at C.U.P.M.C. Meeting,
Sheridan Park, Ont., May 20th, 1983.

Title:	Management of Radionuclides in Uranium Mill Tailings
Period of Award:	Officially Nov. 1, 1978 to Oct. 31, 1981. Actual period was from March 1979 (notification date) to May, 1982.
Original Grantees:	S. Banerjee (principal grantee) M.H.I. Baird A. Corsini A.E. Hamielec
Amounts Awarded:	1978-79: $ 99,000 1979-80: $109,000 1980-81: $135,000

Additional funding of $10,000 was provided as a contract from the Department of Supply and Services in May-June, 1982.

Original Objectives (application made in April 1978)

To contribute to the development of technology which would alleviate the problems of radionuclides in uranium mill tailings. This general objective was to be met by the following means:

(1) Extend technology for removal of radium from sulphuric acid leach tailings.

(2) As in (1) but applied to radium and thorium.

(3) Study encapsulation and sealing methods for mill tailings.

(4) Extend technology for alternative milling processes (e.g. nitric acid leaching) which would simultaneously extract uranium and radium from the ore.

Development of Project

In December 1978, a visiting appraisal team (nominated by N.S.E.R.C.) advised against pursuing objective (3). In July 1980, our principal grantee S. Banerjee ceased to be a full-time faculty member. Although he continued to retain overall responsibility for the project, M.H.I. Baird became the on-site coordinator. Dr. Hamielec withdrew from the group, but it was joined at this time by several new faculty colleagues and from summer 1980 to summer 1982 the faculty responsibilities were as follows:

S. Banerjee (part-time faculty member):	Overall coordinator and principal grantee
M.H.I. Baird:	On-site coordinator and $FeCl_3$ leaching (see below)
A. Corsini:	EDTA leaching (see below)
D.R. Woods:	Selective flotation (see below)
S. Vijayan (part-time faculty member, with A.E.C.L. Pinawa):	General advice and interaction with D.R. Woods
L.W. Shemilt:	General advice on nuclear wastes and ion exchange technology.
K.D. Hester (part-time faculty member, with Rio Algom Ltd.):	General advice on uranium milling technology and management of tailings
M. Tsezos (since Jan. 1981):	Removal of Ra from leach liquors using biomass (see below)

The following salaried research personnel were involved:

A. Mixon (EDTA leaching)
I. Mirdosh ($FeCl_3$ leaching)
S. Muthuswami (Flotation, etc.)
D. Keller (EDTA leaching, analytical methods, biomass, etc.)
A. Pidruczny (analytical methods)
A. Molgaard (biomass)
L. Kulcher (flotation)

plus temporary people employed for very short terms.

Results

Objective (1) of the original proposal was based on preliminary work at McMaster [1] and elsewhere, on the removal of radionuclides from mill tailings by contact with various leachants. It soon became apparent that organic complexing agents such as EDTA (ethylene diamine tetraacetic acid) were much more effective for radium removal if other metals such as calcium and iron were excluded. A. Corsini and his group developed a process in which the sulphuric acid-leached solids were washed with water but not neutralised with lime, then subjected to EDTA leaching. This was found to remove about 85% of the radium from the solids [2]. Moreover, most of the EDTA can be recovered for further use by precipitation from solution at pH 1.5 [3].

In regard to objective (2) of the original proposal, it was found that special analytical techniques for thorium 230, as distinct from the much more abundant 232 isotope, would be required. As the project progressed, it was decided to concentrate on radium removal rather than diffuse the effort by developing the new techniques for 230-Th.

As advised by the visiting committee in 1978, objective (3) of the original proposal was not pursued.

Objective (4) has been followed extensively. At the suggestion of K.D. Hester, the possibility of separating the uranium and radium-rich fractions of the ore using mineral flotation with special reagents has been investigated by a group led by D.R. Woods and S. Vijayan [4]. Certain novel flotation reagents have been found very effective but reagent consumption has been high. Fundamental surface chemistry studies involving uranium minerals and these flotation reagents have been carried out [5]. M.H.I. Baird's group has studied simultaneous leeaching of uranium and radium from ores. Ferric chloride has been found to be an extremely effective leachant, provided sulphides are first removed from the ore by flotation [6]. A small proportion of radium remains unextracted by ferric chloride and the retention mechanism has been studied [7].

Radium can thus be leached effectively by EDTA and by $FeCl_3$, but the question remains as to the best method for recovery of the radium in a concentrated form for disposal.

M. Tsezos has led a group studying the uptake of radium by adsorption on solid biomass. Results are promising for the uptake of radium from simple aqueous solutions [8] and work is in progress on uptake in presence of other cations.

An interim summary of much of the above work [2,3,4,6,8] was presented by M.H.I. Baird at an N.E.A./O.E.C.D. Workshop on Uranium Mine Tailings [9], held in October 1981.

Between September 1981 and April 1982, faculty members and research staff in the N.S.E.R.C. funded group supervised and acted as consultants in a design project with a team of 12 final year chemical engineering undergraduates [10]. The objective of this project was to design and cost three uranium mills based on 5000 t/d of ore, using (a) conventional sulphuric acid leaching, (b) sulphuric acid leach followed by EDTA treatment to remove radium and (c) ferric chloride leach process in which uranium and radium are simultaneously extracted. The principal conclusions from this study based on the N.S.E.R.C. funded research are: (i) the EDTA process, (b) adds about 50% to the cost of conventional milling and (iii) the ferric chloride process appears to be similar in cost to conventional milling, but a few of the design parameters still remain uncertain. The results were presented in April 1982 at a full-day session at McMaster to which representatives of the uranium community were invited. The designs and cost comparisons are compiled in a detailed typewritten report [10].

Two of the technical papers, dealing with EDTA leaching [2] and ferric chloride leaching [6] have been presented at the CIM Conference in Toronto on 29 August - 1 September 1982, and U.S. patent coverage is being sought for certain aspects of the ferric chloride leaching [11].

Benefits Derived from this Project (1978-82)

The main benefits were the technical contributions made to the methodologies for reducing radium levels in the tailings as discharged from uranium mills. The specific details have been or will be published [2-11] and brought to the attention of the Canadian uranium community through visits, workshop and conference presentations.

Faculty members involved in the project benefitted greatly from the interdisciplinary contact between chemists, chemical engineers and engineering physicists. It must be put on record however that the response of outside organizations and individuals to the project was mixed; while many showed interest and cooperation and made useful constructive criticisms, others were apathetic. Nevertheless it is felt that the long-term benefits of the work at McMaster will eventually become apparent after the publication of all our results [2-11].

The research staff employed at McMaster University on this project all developed greatly as professionals, despite (or perhaps partly because of) the occasional setbacks. Dr. Nirdosh is now a faculty member at Lakehead University and is continuing with research on radium removal. Dr. Nixon has accepted a research position with Eldorado Nuclear Ltd. Mr. Keller and Dr. Muthuswami are still employed here.

During 1981-82, the undergraduate chemical engineering design project at McMaster University was greatly enriched by contact with members of the N.S.E.R.C. funded team, and the research results also formed the basis for design projects in the 1982-83 academic year. These projects benefitted from interactions with K.D. Hester (Hester Associates) and Dr. V. Lakshmanan (O.R.F.).

Continuation Beyond July 1st 1982

Several projects relating to uranium mill tailings have continued actively since the termination of the 1978-81 strategic grant.

In the area of radium removal from solution by biomass, Dr. M. Tsezos has continued to lead a small research group. In association with Dr. M.H.I. Baird and Dr. L.W. Shemilt, he was awarded a two-year strategic grant of $78,000 in November 1982. The objectives are to study equilibria and kinetics of radium adsorption on selected types of biomass, with particular emphasis on the effects of competing ions. The elution behaviour of the biomass is also under study.

Dr. D.R. Woods in association with Dr. S. Vijayan (A.E.C.L. Pinawa) is conducting a program on the selective flotation of uranium minerals by chelating agents [12]. The emphasis is on the adsorption mechanism of these agents as this affects flotation reagent consumption.

The main centre of activity on ferric chloride leaching has shifted to Lakehead University since Dr. Nirdosh transferred there in the fall of 1982. He has been considering the various possible mechanisms of radium retention on solids (see ref. [7]) and carrying out experimental work on leaching and retention.

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2. Nixon, A., D. Keller, K. Fritze, A. Pidruczny and A. Corsini, "Radium Removal from Elliot Lake Uranium Mill Solids by EDTA Leaching", Hydrometallurgy (in press, 1983).

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10. Baird, M.H.I., T.W. Hoffman, D.R. Woods, M. Tsezos, J. Brooks, L. Segall, S.H. Noh, S. Muthuswami, A. Nixon, D. Keller, T. Brown, A. Car, J. Chan, D. Darragh, F. Fusca, R. Jones, D. Millar, B. Monahan, B. O'Reilly, P. Penstone, D. Whiffen and P. Zabrok, "Design of Uranium Milling Processes", McMaster University, Chemical Engineering Dept. (May, 1982).

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12. Muthuswami, S.V., S. Vijayan, D.R. Woods and S. Banerjee, "Flotation of Uranium Ores in Canada: Screening Studies of Collectors for Elliot Lake Ores". Dept. of Chemical Engineering, McMaster University, Hamilton (1983).

Discussion of Paper by M. Baird

L. Melis	1) There is a difficulty at CANMET in leaching high arsenical ores from Saskatchewan. What would your process do for those ores?
	2) Is there any concern for chloride in the effluents?
M. Baird	1) No funding is available for testing the ores.
	2) Yes. Chloride content in the effluents is a concern.
C. Lendrum	Which solutions are treated for radium?
M. Baird	The pachuca discharge is treated for radium removal.

Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

CANMET
Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

URANIUM FLOWSHEETS OF THE FUTURE

G.M. Ritcey
Extractive Metallurgy Laboratory

January 1984

Presented at 1983 meeting of Canadian Uranium Producers' Metallurgical Committee, Toronto, Ontario, May 1983 and for publication in the Proceedings of the CUPMC.

Project ERP-4.4.1.0.01
Uranium Extraction Technology

MINERALS RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES
DIVISION REPORT ERP/MSL 84-10 (OP&J) DRAFT

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INTRODUCTION

The H_2SO_4 leach process for treating uranium ores followed by the remainder of the conventional flowsheet has been in use for over 30 years throughout the world. That process, as well as the alkaline leach process, which has been installed in only a few plants, have proven acceptable in that the desired final uranium product has been successfully produced. However during the past decade there has been increasing concern for the environmental aspects of processing. This concern for the environment and the subsequent research by investigators worldwide has resulted in improvements to the process, mainly in the disposal and management of the tailing.

During the leaching process, radionuclides of concern (Ra^{226}, Pb^{210}, Po^{210}, Th^{230}) remain essentially in the tailings residue. The little that remains solubilized in the aqueous effluent is precipitated as a $BaRaSO_4$ sludge with $BaCl_2$ which slowly settles in a series of tailing ponds. Thus the discharge effluent from the final pond meets acceptable federal criteria of 10 pCi/L or less of Ra^{226}. The solid tailings, through the process of weathering and time, release radionuclides to the water system and thus could constitute a problem in the future. In addition, as most uranium ores in Canada contain appreciable amounts of sulphide, the weathering process oxidizes the sulphide to sulphuric acid. This acid can, in addition to providing an acid effluent, also dissolve other contaminants in the tailings. Therefore the uranium tailings can result in the discharge of radionuclides and other contaminants to the environment. In only one plant, using HNO_3, would there be the possibility of significantly reducing the amount of radionuclide release to the environment. Such a process solubilizes most of the radionuclides which could be subsequently isolated in the mill.

With primary consideration being given to the environmental aspects, CANMET began a research program in 1977 designed to improve existing technology as well as to develop new alternative technology. The objective was to reduce the environmental impact, improve by-product recovery, and to maintain an economical viable process.

If we are to consider the development of alternative technology, then the technology selected must possess many advantages. Such rationale are presented in Table 1. Obviously it would not be necessary to fulfill all the stated requirements, but if the environmental problems and uranium recovery are satisfied then any other benefits would be additional.

Table 1. Emphasis and rationale for considering alternative flowsheets

- Decrease the environmental impact
- Isolate radium in mill
- Increase the uranium recovery (95-98%)
- Increase the possibility of by-product recovery (As,Ni,Th,RE's)
- Decrease the capital and operating costs (eg. liquid/solids separation)
- Decrease energy requirements
- Increase solution recycle
- Decrease acid drainage in tailings
- Decrease overall tailings maintenance

This paper discusses and compares some of the flowsheets resulting from research that has been ongoing at CANMET during the past 5 years on improvements in the uranium processing flowsheet. A possible flowsheet is proposed for the future.

Flowsheet Comparisons

Conventional H_2SO_4 process

In the period of 30 years in which the conventional H_2SO_4 process has been used, no major break-throughs have occurred in the process, and so improvements have been incremental both in economics and in uranium recovery. Thus we have seen developments in each of the unit processes, as shown in Fig. 1.

Autogenous grinding has been introduced and preconcentration might prove feasible for some ores for the removal of sulphides or radium.

Leaching is still with pachucas or tanks with some plants having gone to autoclaves. The oxidation route, with some possible cost reduction, can be achieved by use of Caro's acid, H_2O_2, O_2 and with less problems with the use of $NaClO_3$. The expensive liquid-solids separation stage still accounts for the major capital and operating costs in the mill circuit. The purification circuit has not changed much in the 30 years, being either ion exchange or solvent extraction. Some resin in pulp plants have operated, and continuous ion exchange is now increasing in popularity. Solvent

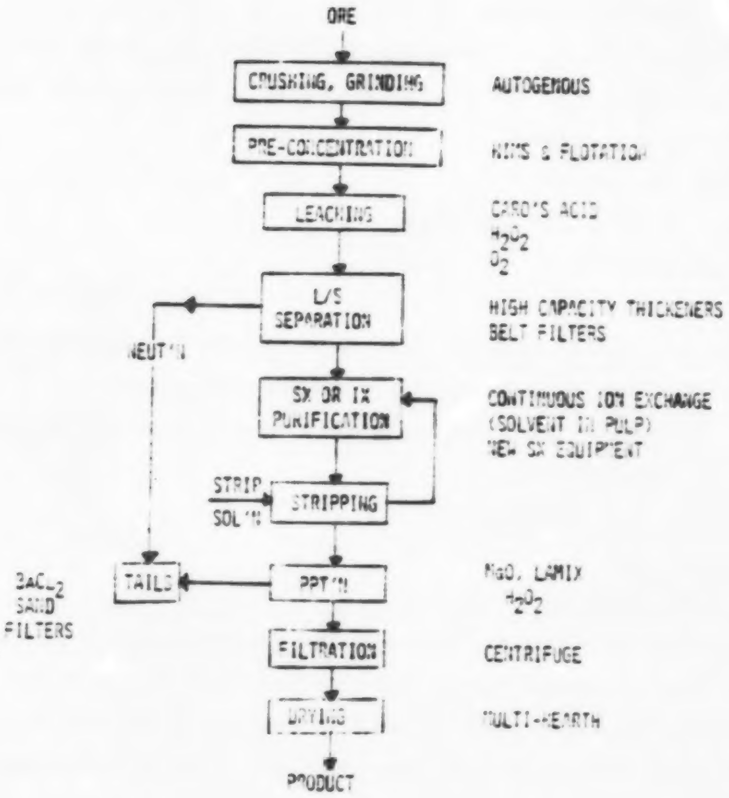
ORE
CRUSHING, GRINDING
AUTOGENOUS
PRE-CONCENTRATION
HIMS & FLOTATION
LEACHING
CARO'S ACID
H_2O_2
O_2
L/S SEPARATION
HIGH CAPACITY THICKENERS
BELT FILTERS
NEUT'N
SX OR IX PURIFICATION
CONTINUOUS ION EXCHANGE
(SOLVENT IN PULP)
NEW SX EQUIPMENT
STRIP SOL'N
STRIPPING
$BaCl_2$
SAND FILTERS
TAILS
PPT'N
MgO, LAMIX
H_2O_2
FILTRATION
CENTRIFUGE
DRYING
MULTI-HEARTH
PRODUCT

Figure 1. Conventional Flowsheet and Incremental Improvements.

extraction has had the greater developments, with many new contactors, improved understanding in the running of the circuit and in improving the quality of the strip solutions. Although solvent in pulp has been piloted, no plants have been built in the Western World. Precipitation has changed only slightly from the first caustic precipitates to ammonium, then MgO, LAMIX and H_2O_2. The peroxide route produces good quality precipitates that filter and settle well resulting in high purity. The original filtration and drying using drum filter and kilns have been improved, in some situations with the incorporation of centrifuges and multi-hearth furnaces. In the tailings treatment, instead of only a lime neutralization, now $BaCl_2$, and in some plants, sand filters have been introduced to maintain a low radium content in the effluent flow from the tailings.

Before examining recent research and considering alternatives to the present process, we must first look at the advantage and disadvantage of the present technology. Table 2 notes some of those major factors, the main disadvantage being the aspect of tailing management.

Table 2. Conventional flowsheet (H_2SO_4)

Advantages

Proven over time

Produces acceptable uranium product for further refining

Some by-products are possible

Disadvantages

High cost L/S separation (capital & operating)

High cost tailings management

400 pCi/g Ra^{226} in tails

No solution recycle (essentially no provision)

High lime requirements for neutralization

Chlorination roast-leach process

In an attempt to decrease the quantity of radionuclides going to tailings the conventional process would have to be modified in some way or changed completely. Because H_2SO_4 does not solubilize the radionuclides, other circuits were investigated. By using HNO_3, although most of the radionuclides were solubilized when treating ore containing 0.1% U_3O_8, nevertheless the tailings still contained about 100 pCi/g Ra^{226} (1). Further studies included the possibility of using the chloride ion to provide radionuclides solubility. Two routes were concurrently developed; one a chlorination roast followed by leaching, the other a leach in HCl.

The chlorination roast flowsheet is shown in Fig. 2 (2). A two-stage chlorination is used, 400°C to volatilize the uranium chloride, followed by 600°C to convert the radium to a chloride. This residue is then leached in dilute HCl to solubilize the radium chloride, as well as any uranium, thorium and rare earths that may be present. This solution is then treated to remove and isolate the radium for burial and the solution bleed to the uranium and thorium recovery circuit. The major portion of uranium from chlorination would be condensed and the condensate purified to recover uranium, thorium and remove iron. Chlorine would be recovered in the process to be recycled to the roasting stage. The process has been demonstrated in a small continuous shaft furnace of 0.5-1.0 kg/h capacity, and residues containing 15-20 pCi/g Ra^{226} were produced. In Table 3 the advantages and disadvantages of such a process are noted. Although the process has many potentially good features, there is much yet to be proven on an integrated circuit at a sufficiently large scale to verify the process. The engineering and design of the reactor, and possibly the treatment of the off-gases and the recovery of chlorine, may be a major obstacles.

Hydrochloric acid leach process

In the hydrochloric acid leach process, shown in Fig. 3, the ore is leached in HCl with the addition of sodium chlorate as an oxidant (3). Either a one or two-stage leach at 70-80°C is employed, depending upon the ore, giving a final tails containing about 10 pCi/g Ra^{226}. The leach filtrate, containing the U, Th, Ra is then sent for purification. First the uranium would be recovered by solvent extraction using trioctyl-phosphine oxide, stripping with H_2SO_4 + H_2O_2 mixture and subsequent precipitation of uranyl peroxide with H_2O_2 (4). The thorium would then be recovered by solvent extraction (5) and the radium removed and isolated on an ion exchange resin. The solution is returned to leaching, with the addition of HCl and $NaClO_3$. The advantages and disadvantages of such a process are given in Table 4. This route appears to be more promising than the chlorination route and is more closely allied

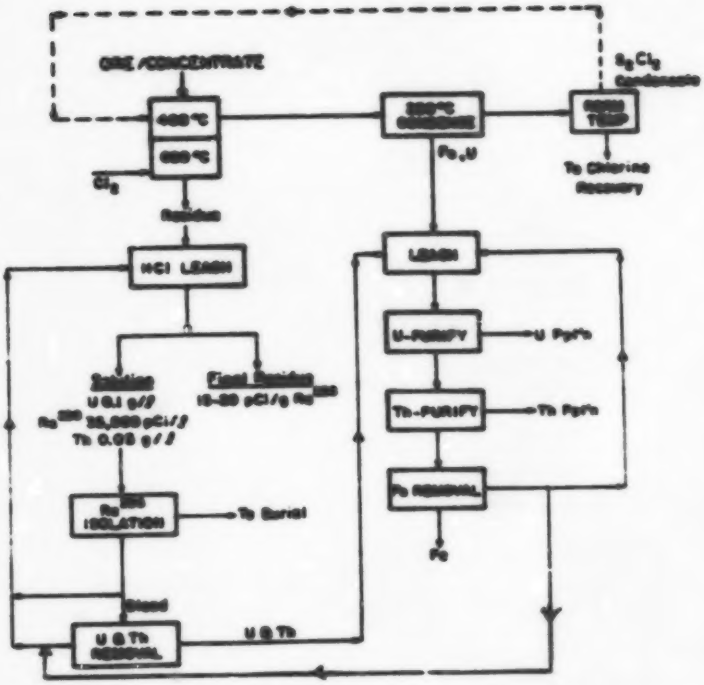
ORE/CONCENTRATE
Cl_2
Residue
HCl LEACH
Solution
U 0.1 g/l
Ra^{226} 35,000 pCi/l
Th 0.05 g/l
Final Residue
10-20 pCi/g Ra^{226}
Ra^{226} ISOLATION
To Burial
Bleed
U & Th REMOVAL
U & Th
S_2Cl_2 Condensate
To Chlorine Recovery
Fe, U
LEACH
U-PURIFY
U Ppt'n
Th-PURIFY
Th Ppt'n
Fe REMOVAL
Fe

Figure 2. Chlorination Roast Flowsheet.

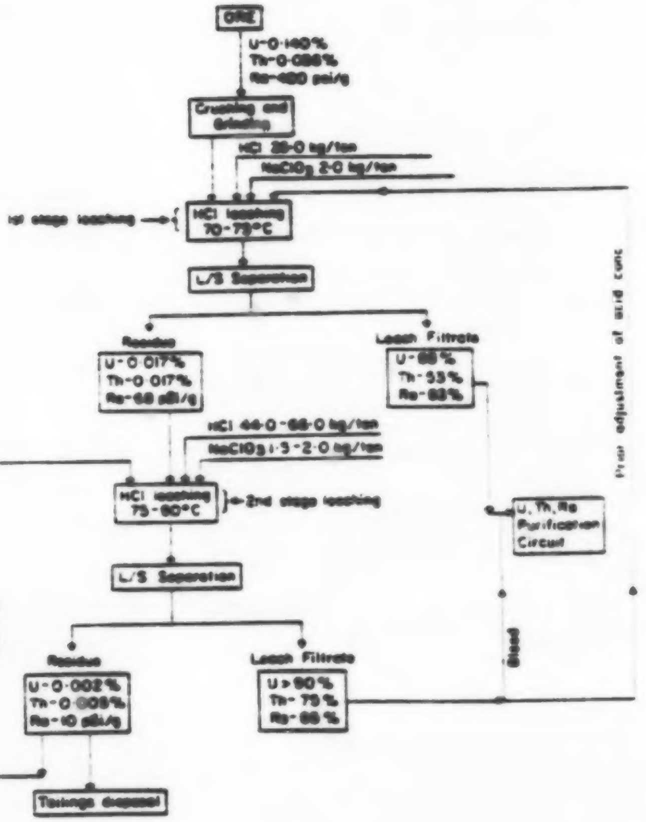
ORE
U-0·140%
Th-0·086%
Ra-480 pCi/g
Crushing and Grinding
HCl 28·0 kg/ton
NaClO3 2·0 kg/ton
1st stage leaching
HCl leaching 70-75°C
L/S Separation
Residue
U-0·017%
Th-0·017%
Ra-68 pCi/g
Leach Filtrate
U-88%
Th-53%
Ra-83%
Prior adjustment of acid conc
HCl 44·0-66·0 kg/ton
NaClO3 1·5-2·0 kg/ton
HCl leaching 75-80°C
2nd stage leaching
U, Th, Ra Purification Circuit
L/S Separation
Washing
Residue
U-0·002%
Th-0·008%
Ra-10 pCi/g
Leach Filtrate
U>90%
Th-75%
Ra-86%
Bleed
Tailings disposal

Figure 3. Hydrochloric Acid Leach Flowsheet.

Table 3. Chlorination

Advantages

- 25 pCi/g Ra^{226} in tails on low grade ores
- Increased by-product recovery possible
- Decreased sulphides in tails
- Much of solution recycled
- High purity uranium product possible
- Ra^{226} isolation in mill
- Increased uranium recovery

Disadvantages

- Cl_2 recovery necessary for economics (good engineering required)
- Higher maintenance likely
- Unproven on complex ores (200 pCi/g Ra^{226} the lowest achieved)
- High costs on low grade ores
- Integrated circuit has not been run

Table 4. HCl leaching

Advantages

- 25 pCi/g Ra^{226} possible (low grade ores)
- Decreased tailings maintenance
- Ra^{226} isolation in mill
- Increased by-product recovery possible (Th, RE's)
- Increased uranium recovery
- Economics for low grade ores seem attractive
- Minimum solution to tailings
- Minimum neutralization required

Disadvantages

- Not proven on complex ores (200 pCi/g Ra^{226})
- Higher maintenance likely
- Continuous integrated circuit not demonstrated on large scale
- HCl must be recovered for recycled

to the conventional technology in the aspects of equipment types. It has been successfully demonstrated in a small continuous pilot plant and the costs (6) are estimated to be approximately equal to that of the conventional route for the treatment of low grade ores.

New H_2SO_4 Flowsheet

In the chloride-based processes there are both good and bad features, even though the environmental aspect of achieving low radionuclide content in the tails can be obtained. Consideration should be given to the possibility of modifying the present conventional H_2SO_4 flowsheet. The following is a proposed scheme to treat the uranium ores by a slightly modified and improved process. This is shown in Fig. 4.

The ore is slurried in water and is pressure-leached under an oxidizing condition to convert the sulphides to sulphuric acid to leach the uranium. During the leach the radium is complexed so that it is rendered soluble. Following leaching, the uranium is extracted from the slurry in a solvent-in-pulp process (7) and the solids passed to a wash thickener to recover any possible values. This wash solution is returned to leaching and a bleed stream of the original thickener overflow after uranium extraction is treated to control build-up of impurities before returning to the leaching stage. The gypsum from the treatment stage is blended with the thickner underflow solids and repulped in brine or dilute HCl to solubilize Ra^{226} which is subsequently isolated for disposal. The thickener underflow solids are water-washed before final discharge to tailings and the wash water returned to the chloride circuit.

The extracted uranium is stripped by a novel technique (8) and the uranium hydrolysed, enabling recycle of the strip reagent. Following filtration the product is dried to produce a high purity compound. The process has many advantages over any of the other flowsheets described. Table 5 lists these advantages. Although each of the segments of the flowsheet have been individually tested, an integrated circuit needs to be run in order to obtain mass and solution balances.

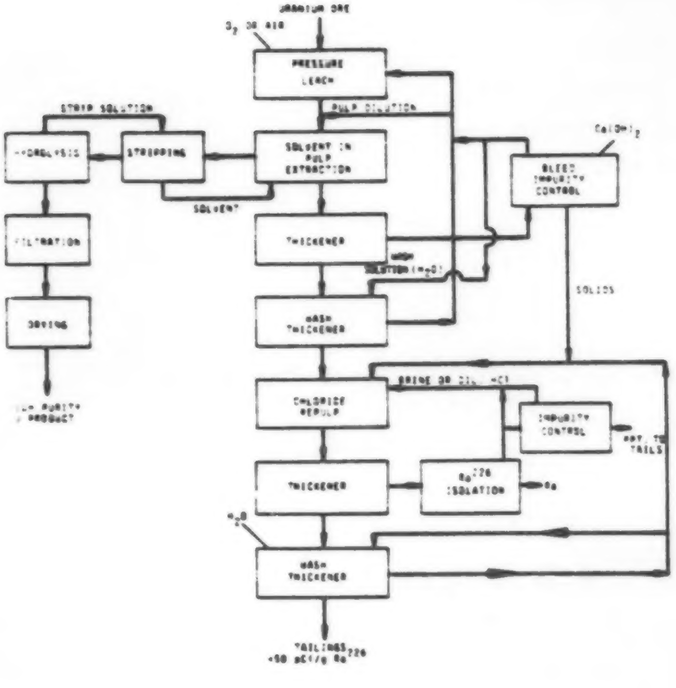
URANIUM ORE
O_2 OR AIR
PRESSURE LEACH
PULP DILUTION
STRIP SOLUTION
HYDROLYSIS
STRIPPING
SOLVENT IN PULP EXTRACTION
$Ca(OH)_2$
BLEED IMPURITY CONTROL
SOLVENT
FILTRATION
THICKENER
WASH SOLUTION (H_2O)
DRYING
WASH THICKENER
SOLIDS
HIGH PURITY U PRODUCT
BRINE OR DIL. HCl
CHLORIDE REPULP
IMPURITY CONTROL
PPT. TO TAILS
THICKENER
Ra^{226} ISOLATION
Ra
H_2O
WASH THICKENER
TAILINGS
<50 pCi/g Ra^{226}

Figure 4. Proposed Modification of Conventional H_2SO_4 Flowsheet.

Table 5. New H_2SO_4 flowsheet

Advantages

Smaller plant

Decreased L/S separation costs (operating & capital)

Higher purity product

Increased uranium recovery (decreased soluble losses)

Increased by-product recovery possible

Decreased energy costs

Decreased tailings management

Maximum solution recycle

Smaller tailings disposal areas

Minimum of lime requirements

$<$50 pCi/g Ra^{226} in tails

Isolation of Ra^{226}

Minimum of sulphide in tails

Disadvantages

Flowsheet not proven on integrated circuit (low grade ores)

Not proven on complex ores

Discussion and Conclusion

Although several possible flowsheets for future uranium mills have been indicated there may nevertheless still be a point for argument and discussion. That is, the flowsheets described are based on the premise that a circuit has to have the radionuclides removed in the mill rather than having the radionuclides disposal in the tailings. Thus the radionuclides would be isolated for subsequent disposal.

But we must consider the two scenarios of isolation in the mill as opposed to the disposal in the tailings. With the latter route as presently practised, the radionuclides gradually seep out over time. The amount or concentration in the effluent is maintained small by proper control of the $BaCl_2$ addition. Thus the tailings disposal areas are presently well designed and continually being improved with respect to their stability - but at a considerable cost.

However the alternative flowsheets that have been prepared do produce tailings containing essentially no radionuclides and therefore much of the tailings management costs would be reduced. But there may well be considerable problems in the isolation of the radium in a compact concentrated form. The objective would be to ultimately dispose of the concentrated radium as a high level waste. At the present time no technology has been developed to provide for a high grade concentrate and no work has been done on the investigation of possible burial of such a material. Discussions have been held with AECL personnel regarding the incorporation of the isolated radionuclides into glasses for burial with the high level wastes.

The approximate amount of radium^{226}, based on 7000 t/d would be 2.8 curies Ra^{226}/d or 365 x 2.8 = 1022 curies/a.

Regardless of the concern about the radionuclides - and many people question whether we do in fact have problems with radionuclides in the current technology - there would be advantages in increasing solution recycle, decreasing solid-liquid separation costs, improving the recovery and the grade of product and increasing by-product recovery. Increased research, if successful, could result in the achievment of such advantages which would benefit the mill and the environment.

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Discussion of Paper by G. Ritcey

K. Hester At the end of the circuit the slurry is about 55% solids and would go to belt filters. The question is whether solvent in pulp (SIP) is technically feasible or whether more work is required before it is possibly operative.

G. Ritcey The SIP work at CANMET was stopped about 10 years ago about the same time as Rio Algom stopped their work on the process. On leach slurries of the Elliot Lake material containing about 30% solids we encountered losses of solvent of about 1 lb/tonne dry feed ore. The adsorption losses were high on pyrite, muscovite and quartz. Such losses were not tolerable and subsequent research showed that the addition of a surfactant, fish glue, to the slurry reduced the losses to 0.06 lb/tonne. This value was considered acceptable. Since then we worked on other aspects of solvent extraction including emulsions and cruds and loss of organics on gangue. The shear or power input to the system and the chemistry are major contributors to cruds. That is the presence of such cations as Ca Mg Al Si Fe can contribute to emulsions and crud. Also we have had some experience with the treatment of tar sands, so that now we believe we are in a better position than before to optimize the SIP process.

C. Lendrum Are any plants using heavy resins?

G. Ritcey Although heavy resins have been developed and are on the market, I am not certain of their use commercially. Of course, Canmet and others have done research on resin-in-pulp.

C. Lendrum Are there any plants using Caro's acid in the leaching stage?

G. Ritcey Yes, the Queensland Mines Nabarlek operation in the Northern Territory of Australia are successfully using Caro's acid.

C. Farrow The flowsheet shows two stages of wash thickeners following solvent in pulp. My questions are:

1) why don't you take the thickener overflow and use SX and IX to remove the radium?

2) with the two stages, I don't see that any savings would result in the liquid/solids separation. You exchange organic losses for aqueous losses.

Discussion of Paper by G. Ritcey (cont'd)

G. Ritcey	The overflow is just the acid sulphate solution carrying some dissolved radium which is recycled to leaching. If the sulphate was not removed, then on reaching the brine wash circuit for radium removal, radium sulphate will be precipitated instead of radium chloride being solubilized. As to the economics of eliminating the liquid-solids separation stage, I believe that considerable savings in capital and operating costs will result, and because SIP removes all uranium from solution there would be increased uranium recovery. Presently the liquid-solids separation stages can account for 2-5% of soluble losses of uranium.

LONG TERM PLANS FOR URANIUM TAILINGS RESEARCH

by

V.A. Haw
CANMET

National Uranium Tailings Program

(Paper to be Given at the Meeting in Toronto of the
Canadian Uranium Producers Metallurgical Committee)

Title: The National Uranium Tailings Program -- Long-Term Plans

Two years ago I spoke to you about the National Uranium Tailings Program, thinking at the time that we would be in operation within a few months, so much for wishful thinking. It was not until last September that the Minister of State for Mines, Mrs. Erola, announced the Program and at the same time released the report of the National Planning Group, which contained the conclusions and recommendations on the need for a research program on uranium tailings management. The Treasury Board approved the terms and conditions for the Program last November, at which time it officially came into being. As many of you know by now, the Program has been cut back substantially in size from that proposed in the National Planning Group Report. The resources approved by the Treasury Board were nine and one-half million dollars and a staff of seven. The duration of the Program is to be five years.

It is my purpose this afternoon to give you a progress report on our activities to date and tell you something of our plans for the future. First of all, I am pleased to tell you that Roy John and Grant Feasby have joined our staff as project managers. I have no doubt that they are both well known to many of you, who will realize that we are fortunate indeed in obtaining their services. For modelling expertise we are considering other alternatives, possibly AECL or a private consultant to assign someone with experience to the Program, which will complete our professional staffing requirements.

Apart from our Program management activities, the Program is to be entirely contracted out. Last November, with the authority to get the Program underway, I took action to place a number of contracts that I thought would be required regardless of what direction the work might take in the future. These have now been virtually all completed and I will describe them briefly to you at this point.

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1. Production of reference material standards from four uranium tailings sites in Canada.

For this contract samples from the four tailings sites were sent to a number of analytical laboratories and analyses were carried out for both the radioactive and a number of non-radioactive constituents. Four bulk samples were obtained from: Madawaska, Rio Algom, Beaverlodge - Eldorado, and Rabbit Lake. The radioactive determinations were made for ^{230}Th, ^{226}Ra, ^{210}Pb and ^{210}Po; for the uranium/thorium ores the following elements were also determined ^{232}Th, ^{228}Ra, ^{228}Th. Other elements determined for these samples were: iron, titanium, aluminum, calcium, barium, uranium, thorium, total sulphur and sulphate.

2. The development and generation of a uranium tailings data base.

With the very large amount of chemical and physical data currently on hand on uranium tailings, and that which is expected to be generated over the next few years in the Program, it is essential that we are able to store and retrieve it quickly for a variety of purposes, including our modelling activities. Systemhouse Limited has undertaken this contract for us and we are now at the point where we have a certain amount of data in the computer files in our system 2000 at Ottawa, and for those interested we expect to put on a workshop to demonstrate the operation of the system and program for storing data some time in June.

3. Preparation of a manual for sampling of a mine waste disposal area.

The requirement for this purpose was to prepare an authoritative manual for the definition and implementation of a sampling program on a waste disposal area, i.e., a tailings pond, describing the factors that must be considered in defining the sampling program and the sampling technology used to collect the necessary aqueous and solid samples. Golder Associates was the contractor for this purpose and we now have the completed manual in our hands. Although it is intended primarily for the use of the Program, we hope to be able to put it in a form suitable for general distribution to the public in the near future.

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4. Review of state-of-the-art of dynamic modelling of uranium tailings

The work requested under this contract was to: evaluate and document the state-of-the-art of computer modelling relevant to the pathways of uranium tailings contaminants. The scope of the work was intended to cover a review of available computer codes and data developed for the evaluation of impacts of radioactive and non-radioactive emissions via various transport mechanisms. Significant pathways for the transport of contaminants from uranium mine/mill tailings were to be outlined. The specific models to be reviewed were: hydrogeological models, chemical models, compartmental models, atmospheric models and systems models. Atomic Energy of Canada Limited at Whiteshell undertook this contract, which has now been completed.

Apart from these contracts, I have taken the view from the start that no Program commitments should be made until the staff are all in place and have had the opportunity to size up the requirement for a uranium tailings research program in Canada. The starting point for this size up is, of course, the National Planning Group's report under the chairmanship of Philip Lapp.

In addition, there is an abundance of literature now on uranium tailings, and some of the contract reports we have now completed will provide additional important input. However, to get off to as quick a start as possible after this initial appraisal of the situation had been completed, I called for two contracts to propose a Program structure that could be used for future detailed planning by our Program management. One was for a broadly based framework to cover all aspects of the Program as envisaged in the Lapp Report for which IEC Beak was the contractor; the second contract was for the purpose of focusing in on what we call the disposal technology component of the Program, which was intended to examine current practice in tailings management in Canada and to recommend areas for research in the light of this up-to-date survey. Lawrence Melis was the author of the report which fulfilled the requirements of this contract.

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I thought it would be useful today to use the Beak Report, in particular, as a basis for outlining the general characteristics of the Program as we see it developing in the future. (Lawrence gave you a resumé of some of the studies he made on our behalf yesterday.) Before doing so I would like to emphasize a few points in regard to the Program as a whole.

Program objectives: In its broadest sense the National Uranium Tailings Program is intended to provide a scientifically creditable information base on which decisions can be made in establishing reasonable criteria for the protection of the environment, and which can also be used to propose practical guidelines for uranium tailings management in the satisfactory close-out of any given tailings site. This statement hinges on the meaning of the term satisfactory close-out and for our purpose here it means that certain conditions must be met.

1) Individual radioactive exposures must be below currently accepted levels, and radioactive and non-radioactive releases must meet appropriate air and water quality regulations;
2) A predictable decline in release rates of contamimants to the open environment, which is ideally monotonic in nature.
3) The principles of ALARA (the acronym for As Low As Reasonably Achievable) both at present and into the long-term future must be adhered to.
4) The particular management strategy or technology proposed for a given site will ideally be of a passive nature and not by design, require continual institutional intervention to ensure that it meets the above requirements.

Turning now to a description of the Program, I am going to draw on the IEC Beak Report, for which Dr. Don Lush is mainly responsible. Two points should be made: at present it is a conceptualized outline of a Program, which must be developed in detail by the Program management, and secondly, it is flexible and, of course, subject to revision as we develop our ideas on how best to achieve our Program objectives.

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There is no doubt that the driving force of the whole Program is the modelling activity. Our efforts will be largely concentrated on determining a means of predicting the future impact of uranium tailings on the natural environment and human society. It is not possible for us to monitor or measure into the future. We can only make predictions (through modelling) based upon those processes occurring today, which we can define, monitor and measure. There will be uncertainty associated with the prediction of any future event. The best tools that we have available at present that allow both the prediction of future events, the quantification of the events and the uncertainty associated with them are mathematically-based probabilistic models. These models are based upon processes and mechanisms that have been operational in the past, are on-going today, and are expected to continue into the foreseeable future. The contributions from the other Program components are essential to the modelling activity; it is accepted by all who have been associated with the conceptual planning of the Program components: modelling, measurement and disposal technology.

The general framework for integrating the three components is illustrated in Figure 1. The modelling activity is central to the Program but is dependent upon the measurements input on the one hand to calibrate and validate the models, and on the management technology options on the other hand in order to evaluate the future consequences of different disposal technologies and relate their performances to costs.

As you will note, the modelling activity proceeds from the source (the tailings) through environmental transport to exposure. Each block in Figure 1 represents a set of sub-models that will mathematically describe the physical and chemical processes inherent to each. Linking them together into an integrated set will be a prime objective of the five-year Program, which will permit the evaluation of specific management options for the close-out tailings in relation to close-out criteria objectives. Experience to-date in modelling pathways of contaminants from uranium tailings has shown the largest areas of uncertainty are in the quantification of source

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term behaviour. The environmental transport component of the study can draw more heavily on literature from a wide variety of study disciplines. Environmental exposure data will be largely derived from the transport modelling. A broad data base already exists relating to human exposure, which the Program can draw on. Figure 2 reflects distribution of effort anticipated in a spatial sense for the Program. Most of the effort will be concentrated in the source area, decreasing with distance from the tailings fence boundary.

The accomplishments of the objectives and goals already described are dependent on a carefully planned schedule of events with clearly defined sub-objectives and milestones to measure progress. Control in the use of time and money in relation to achievement of results as we progress from year one to five will be essential if we are to obtain our objectives. The production of a Program outline designed to meet these needs was the main task assigned to IEC Beak. The results, in effect a Program schedule, are shown in Figure 3. I will attempt to point out only the major features of this schedule that we think are important to the Program as we see it developing.

Program Plan and Schedule

The Program plan as illustrated in Figure 3 shows a large number of coloured boxes, each of them numbered, with vertical and horizontal lines indicating the flow of information as the Program proceeds. The colour coding designates the organizational units within the Program; starting at the top, the orange designates the Senior Review Board, the brown, the Technical Advisory Committee, the blue, the Program Director's office, the green is the measurement, or monitoring, component of the Program, the yellow, the modelling component, and the pink at the bottom is the disposal technology component. Each of the boxes represents a specific activity. Most of them represent a major piece of work which will require detailed planning by the Program staff, including the definition of sub-objectives, allocation of resources and other project management activities. Within

.../7

each of the boxes there are numbers to indicate the starting day into the Program. The vertical lines illustrate the linkage between the Program components to achieve complete integration of all activities.

Both the Technical Advisory Committee and the Senior Review Board have still to be appointed. The Technical Advisory Committee will consist of about twelve to fifteen persons, representing industry, provincial and the federal government. In addition, they will be appointed based upon their individual expertise on various aspects of the Program activities. The Senior Review Board was originally conceived for the purpose of coordinating Federal/Provincial activities in monitoring the progress of the Program, as well as to take overall responsibility for direction of the Program to the respective governments. These terms of reference may change somewhat, but it will still have an executive function in monitoring Program progress.

Blocks of 6 to 15, inclusive, cover the orientation of the staff, the review of the main Program objectives, an agreement on the strategy for Program implementation, and the planning of a detailed schedule for the first year's work.

Boxes 16 to 20 include the preparation of a manual on sampling techniques, techniques, the preparation of chemical standards and decisions on analytical procedures. A review of existing models relevant to uranium tailings management will be undertaken (already completed), and the selection of an appropriate systems model will be made, which will be made on the present state-of-the-art on tailings management, together with an inventory of existing tailings sites in Canada (again, already prepared). This will be preparatory to selection of sites on which detailed studies will be made.

Activities described for boxes 21 to 28 include the creation of a computer data base for uranium tailings and the collection of all available and relevant data to uranium tailings management. This will include the quality evaluation of the existing data.

.../8

It is perhaps worthwhile noting at this point that there will be periodic reviews of Program progress, possibly every three months. These periodic reviews will be made by Program management to the Technical Advisory Committee and on through to the Senior Program Review Board.

The next stage will be to define the sub-models required to satisfy the systems model as shown in box 23. It will be necessary to determine what is now available and relevant for this purpose, and what development of new sub-models will be required to simulate the behaviour and migration of tailings contaminants.

For new model development research will be required to describe the natural processes taking place in which tailings constituents are involved. The selection of sub-models, existing, modified or new, will also determine the field measurements required. It is anticipated that 60 to 70 percent of the field measurements required will be within and at the boundary of the tailings, the balance will be concerned with the pathways outside this area. These activities are related to boxes 24 and 25 in the diagram.

The computer system required to operate the model, and the necessary software, must be assembled. The programs, or codes, for the models must then be de-bugged. These steps are shown in boxes 26 and 27.

At this juncture it is proposed to define a hypothetical tailings site that can serve to test the model. This will be based on data available at this point in time, which can be used for model input. Values will have to be assigned the parameters for the hypothetical site, which will include measurements for both the field data, e.g., chemical composition, PH, oxidation rate, etc., and for the barrier system associated with different management options. In addition, ranges of values will have to assigned that reflect the uncertainty of values measured and the probability distribution of these values. This general area of activity also includes the definition of tailings management options with their associated input parameters. These activities are covered in boxes 29 to 32, inclusive.

.../9

The model is now ready to run using the parameters from the hypothetical site. The sensitive parameters and model sub-components are determined by a series of model sensitivity runs, in which model response is evaluated against each of the parameters. In addition, the models will also be run to incorporate a probabilistic error analysis in order to judge overall confidence in model output, given the wide range of input parameter uncertainties. Special attention will be given to the sensitive parameters for both field data and disposal technology options in order to obtain a high level of accuracy in measurement, and to assure that the theory and algorithms for model sub-components are appropriate. Review and modification of models will be done as required in light of new information from on-going research. Boxes 33 to 39, inclusive, cover these activities.

At this point, the kind of data required will have been identified, as will the level of accuracy required for the measurements for use in the models. Field measurements and other data gathering can now begin. This will demonstrate the sites specific applications of the model, which can now be calibrated and verified. These actions are covered in boxes 40 to 43.

With the identification of sensitive management options, in relation to model output, projects can now be planned to investigate the performance and uncertainty associated with different management options, including barrier systems, milling processes and tailings placement methods. These items are identified in box 44.

Cost estimates will be made for incremental measures taken to reduce the release of contaminants for the experimental sites under study. This information, coupled with model output, will permit application of the ALARA principle, i.e., the optimization of the savings in environmental detriment versus the cost of achieving these savings. This area of work is covered in boxes 45 to 49, inclusive.

.../10

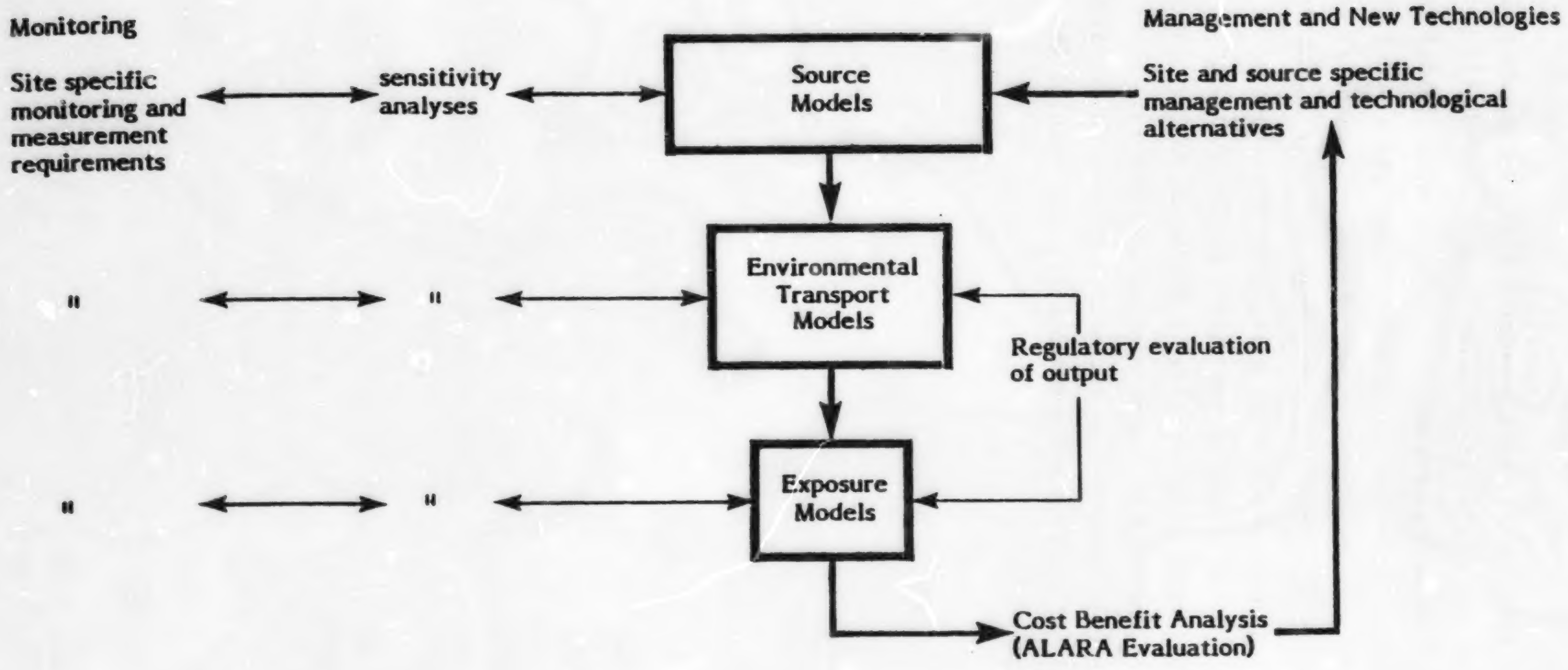
Finally, so far as the Program Plan is concerned, we have arrived at a major decision point at which an interim report will be prepared on the overall Program, describing what has been accomplished and to what degree objectives have been achieved. Reports can also be prepared on the evaluation of the future performance of available disposal technology and on the description of the model, together with user documentation. The report writing and the decisions will, of course, have to be taken on whether the Program should be continued, on the direction it should take, and on what additional information is needed to achieve the overall objectives of the Program. Continued work beyond this point in time is, of course, highly speculative, but would probably be iterative, using additional input data in order to obtain better results and a lesser degree of uncertainty.

Figure [illegible].1.

NTPO

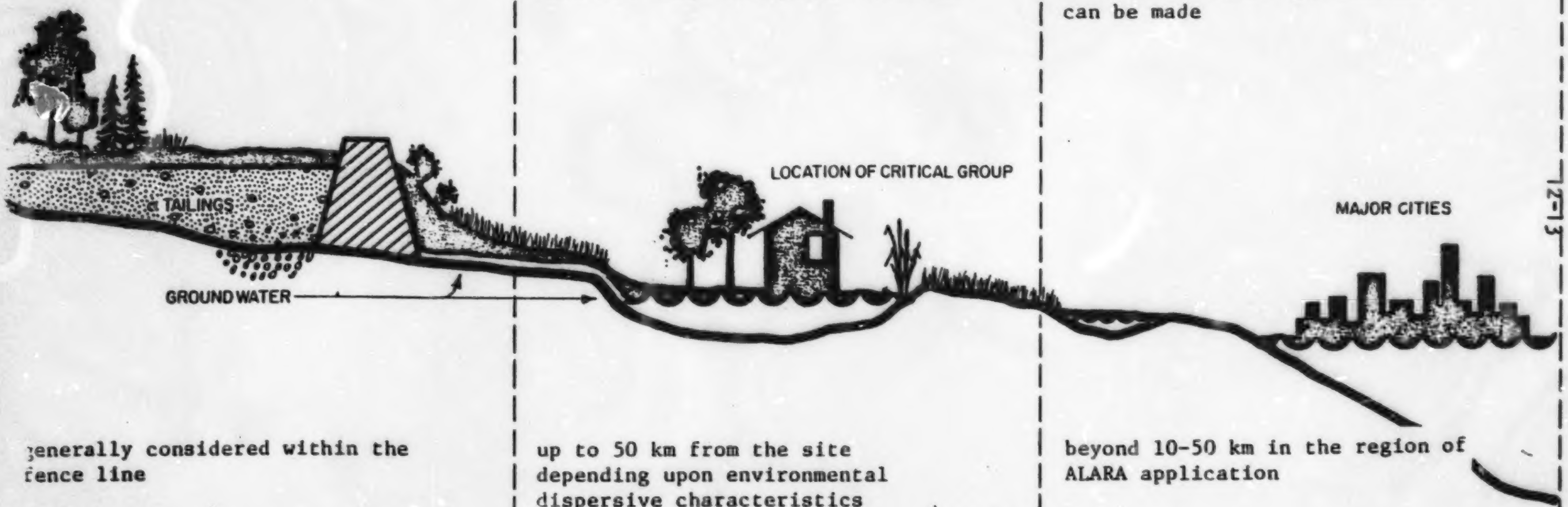
CONCEPTUAL PROGRAM DESIGN

Generic Modelling Framework


Monitoring
Site specific monitoring and measurement requirements
sensitivity analyses
Source Models
Management and New Technologies
Site and source specific management and technological alternatives
"
"
Environmental Transport Models
Regulatory evaluation of output
"
"
Exposure Models
Cost Benefit Analysis (ALARA Evaluation)

Schematic Breakout of Proposed Program Level of Effort

SOURCE AREA	IMMEDIATE RECEIVING ENVIRONMENT	REMOTE RECEIVING ENVIRONMENT
in which disposal technology can be applied	in which monitoring can be carried out and levels of contaminants discernible above background detected	in which meaningful monitoring cannot be carried out but for which theoretical calculations can be made
generally considered within the fence line	up to 50 km from the site depending upon environmental dispersive characteristics	beyond 10-50 km in the region of ALARA application
60%	35%	5%
Responsibility of both disposal technology manager, monitoring manager and modelling manager	Responsibility of monitoring manager and modelling manager	Responsibility of modelling manager



12-13

Under overall coordination of program director

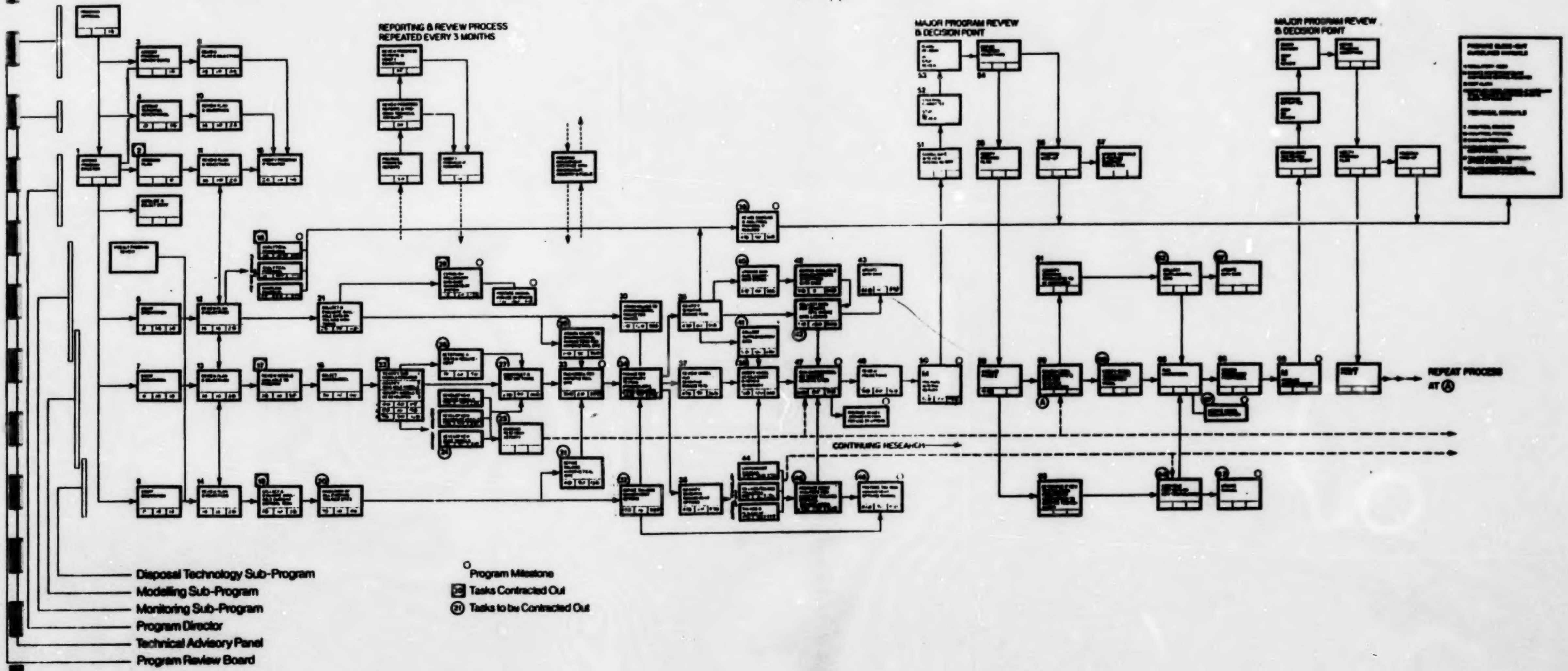
Figure [illegible].2

Figure 3

5.1 Program Plan

Energy Mines
and Resources
(CANMET)

National Tailings
Program Office


REPORTING & REVIEW PROCESS
REPEATED EVERY 3 MONTHS
MAJOR PROGRAM REVIEW
& DECISION POINT
MAJOR PROGRAM REVIEW
& DECISION POINT
CONTINUING RESEARCH
REPEAT PROCESS
AT Ⓐ
Disposal Technology Sub-Program
Modelling Sub-Program
Monitoring Sub-Program
Program Director
Technical Advisory Panel
Program Review Board
○ Program Milestone
Tasks Contracted Out
Tasks to be Contracted Out

Discussion of Paper by V. Haw

L. Heymann What concerns me is whether there is a problem in mine tailings, for two mines in their pathways analyses have found that there is not a great problem.

V. Haw We realize that there is much thought along those lines, but there exists a problem in the minds of the public, and particular groups. So the polititians react. It is a small parallel to the high level waste program. It is a Research Program.

C. Lendrum I thought I might hear something on the review of the various assessment boards which included experts from around the world. All that data have never been analysed and synopsized. Also there are a great many areas in the world (e.g., Johannesburg) where they live on the tailings — and no effects. As long as there is money being spent, people will perceive there is a problem.

V. Haw Whether there is a real problem or not, the object would be to determine if there is one. There is a great sensitivity to anything nuclear, and three provinces have moritoria on uranium mining. We don't intend to ignore the evidence available that could be reviewed.

C. Lendrum (by letter)

The study should not start by assuming there is a problem. The recent minutes of the OECD/NEA Committee concludes uranium mill tailings are a nuisance rather than a hazard.

Therefore this new study (NUTP) should technically review and synopsize the sworn evidence given by world experts at the BAYDA Commission and the Elliot Lake Environmental Assessment Board Hearings. These experts included Drs. R.D. Evans, G.W. Dolphin, Archer, Morgan, Soccamanno, Edwards, as well as most of the uranium metallurgists in Canada, including Al Ross, Ken Hester, Ken Coyne and myself. There probably has never been a greater collection of internationally accepted experts assembled to give sworn evidence on uranium tailings than at the Environmental Assessment Board Hearings at Elliot Lake. The transcript of this evidence is available.

Discussion of Paper by V. Haw (cont'd)

C. Lendrum (cont'd) The BAYDA Commission in Saskatchewan also had a large number of eminent authorities on the subject, both pro and con and their sworn testimoney is available. Both Commissions found in favor of the industry.

These two Commissions cost the governments and the mining companies in the neighborhood of $15,000,000. It is a crime that this vast fund of evidence on the perceived hazards of uranium tailings is not being used as a starting point for this latest investigation.

The development of mathematical models is a serious problem. The Canadian delegation has presented a model to the OECD/NEA Committee which infers the impact of radon from a tailing pile in Elliot Lake impinges on the shores of Europe. This, in spite of the fact you cannot get a reading above background a kilometer away from the tailing area. Compare this with the proximity of tailing piles to the City of Johannesberg, South Africa. These piles are 100 years old. The operators agree with Dr. Costello who says such models "strain credibility".

I agree with Vic Haw that there is a perceived problem but we must work on the problem not become part of it.

ELDORADO RESOURCES LIMITED

BLIND RIVER REFINERY

PRESENTED TO THE C.U.P.M.C.

MAY 20, 1983

J.M. SCHISLER and

D.J. FRANCIS

ELDORADO RESOURCES LIMITED
BLIND RIVER REFINERY

Eldorado Nuclear Limited originally planned to build a 9,000 metric ton UF_6 plant west of Port Hope, at Port Granby. This site was subjected to an environment review panel and was ruled not an acceptable site. Environmental reviews were then carried out at three other potential sites; Hope Township near Port Hope, Dill Township near Sudbury, and Blind River. All sites were approved and ENL selected Hope Township. Preliminary site preparation work started in late 1979. In early 1980, the federal government decided that the Blind River location was more appropriate and Eldorado was directed to construct the refinery in Blind River. A cost review was completed on the Blind River site and this determined that it was more economical to construct an 18,000 metric ton concentrate to UO_3 refinery at Blind River and a 9,000 metric ton UO_3 to UF_6 conversion plant at Port Hope, rather than a combined refinery conversion plant in Blind River.

CONSTRUCTION

Redesign reached a point, by the fall of 1980, to allow site work to commence. The refinery was scheduled to be started up in early 1983. Three months of strikes by the construction trades, in the summer of 1982, delayed the planned start-up until August 1983.

Construction by the EPC contractor, Lummus Canada Incorporated, was completed by June 30th. Outstanding and new construction items were done by another construction company.

Commissioning and start-up of the utilities areas was completed in June. Commissioning of the process areas was completed in August, with the introduction of uranium occurring on August 23rd.

To date, all unit operations have been started up and operated independently; problems identified, and corrected.

This week continuous plant operation commenced with the objective of producing 100 tonnes uranium as UO_3 by the end of September.

PROCESS DESCRIPTION

CONCENTRATE SAMPLING

All uranium concentrates are received by road transport in 55 US gallon drums and off-loaded in the warehouse area. Every drum is sampled, either by auger or falling stream sampling systems, and analysed for uranium and impurities. The material is then stored in drums until analyses are completed and the concentrate is required for processing.

FEED SYSTEM

Concentrates are dumped from drums and sent to ten metric ton hoppers. Calcined and magnesium based feeds are proportioned directly into the leaching circuit. Low levels of ammonia are required in the refinery raffinate, requiring ammonium diuranates to be routed through an indirect propane-fired rotary calciner where it is heated to 400°C to drive off the ammonia. The calciner is designed to handle 4500 metric tons of ammonium diuranates per year. The calcined product is then sent to a drumming station or the feed bins for digestion.

DIGESTION

Concentrates from the feed system are entered into digestion from three bins, simultaneously with nitric acid and uranium-containing recycle streams, to produce an aqueous feed to solvent extraction. This is a semi-continuous operation with two leach tanks running continuously, filling four batch surge tanks. Each of these surge tanks is sized to feed one solvent extraction circuit for eight hours. The parameters controlled in the digestion circuit are uranium content, density, and free acid.

SOLVENT EXTRACTION

The solvent extraction system is composed of two independent trains of Mixco columns. All columns are 3 feet, 6 inches in diameter. The extraction columns are 37 feet high and contain 20 mechanical stages, giving 7 theoretical stages. The scrub and re-extraction columns are 29 feet, 6 inches high and contain 15 mechanical stages, giving 3.5 theoretical stages.

Solvent used is 25% tributyl phosphate (TBP) in kerosene. Sodium carbonate treatment of the solvent removes degredation products and allows the solvent to be re-used. All columns are operated organic continuous and the extraction columns are designed to produce <0.1 grams per liter U in the raffinate.

Variables controlled in the solvent extraction circuit are saturation density, extract density, uranium in raffinate, uranium in strip solvent and scrub column organic to aqueous ratio.

BOILDOWN

OK liquor (uranyl nitrate solution) produced in solvent extraction is fed to a triple effect evaporator system where the uranium is concentrated from 100 grams per liter to 1200 grams per liter. The operating cost savings of using the triple effect evaporator system is estimated at $300,000. per year.

DENITRATION

Uranyl nitrate hexahydrate (UNH) from boildown is gravity fed to 12 continuous stirred pot reactors operating at 350°C. The UNH is thermally decomposed producing UO_3, which is then conveyed to 13 metric ton shipping containers for road transport to Port Hope. The denitration design is based on equipment currently used in Port Hope, but has been improved by the addition of proportional temperature control and a fan in the heating jacket to improve heat distribution. The building design leaves room for four additional denitrators which will be obtained from Port Hope when they shut down their UO_3 operation.

RAFFINATE

The raffinate circuit consists of a three stage evaporator system which recovers nitric acid by the addition of sulphuric acid to the raffinate. Water is also removed in this system, reducing the raffinate volume by a factor of six. The concentrated raffinate is shipped to the mills in Elliot Lake where residual uranium values are recovered and the sulphuric acid content utilized.

NITRIC ACID RECOVERY

This circuit consists of bubble cap absorber and concentrator towers.

Oxides of nitrogen (NO_x) generated during the leach and denitration steps are removed in the absorbers to produce 30% nitric acid. The two acid concentrators are used to upgrade the absorber acid and other weak acid streams to 55% nitric acid for re-use in digestion.

SUMP TREATMENT

All spills of dilute uranium carrying streams are sent to the sump treatment area where the uranium is recovered by precipitation with magnesium oxide. The precipitate is removed on a rotary vacuum filter and redissolved in nitric acid for recycle to digestion. The filtrate is routed to a sump evaporator to reduce the volume. The evaporator bottoms are then sent to raffinate for recovery of nitrate values.

SCRAP RECOVERY

UO_2 scrap received from the pellet fabricator is dissolved in nitric acid in the scrap recovery area, then combined with the solvent extraction feed streams.

EQUIPMENT DECONTAMINATION

In this area, pickling tanks are used to remove contamination from equipment or scrap before they are allowed off site for re-use or disposal.

MAIN OPERATING CONTROL ROOM

All process instrumentation is controlled from a central operating control room utilizing a TDC-2000 with three basic operator stations and a supervisory computer. Plant start-up was carried out without computer control, but the computer will be utilized for process adjustments after start-up.

UTILITIES

The Blind River Refinery produces all utility services except for electricity.

STEAM

The steam generation plant is composed of two boilers, each rated at 50,000 pounds per hour at a maximum pressure of 150 psig. Steam is used for process heating, freeze protection and building heating.

WATER SUPPLY

Water is obtained from wells and treated to produce process water, boiler feed water and potable water. A separate pumping system supplies river water for fire protection. The fire system is composed of two pumps, one diesel operated, each rated at 2500 gallons per minute.

COOLING WATER

Cooling water is provided by a closed system utilizing a four-compartment cooling tower. The system provides 6000 U.S. gallons per minute flow of cooling water.

AIR SUPPLY

There are two compressors available to supply plant air and instrument air, each with a capacity of 1317 cubic meters per hour.

SEWAGE TREATMENT

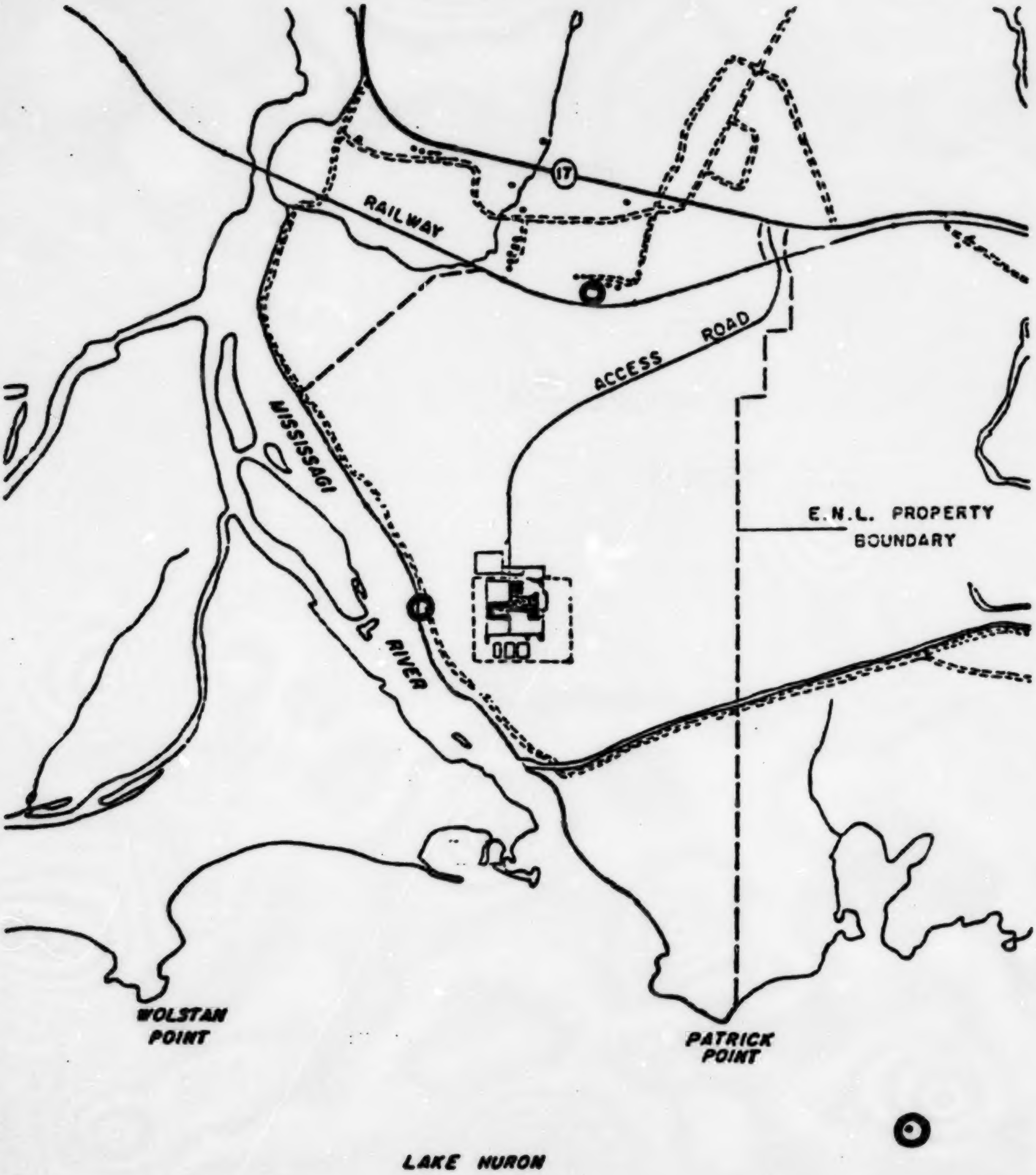
The Refinery has its own sewage treatment plant which is sized to handle 200 people.

EMERGENCY POWER

An 800 kW diesel generator is available to supply electricity in the event of an Ontario Hydro power outage. This generator supplies power to safety related equipment and critical services such as steam and air.

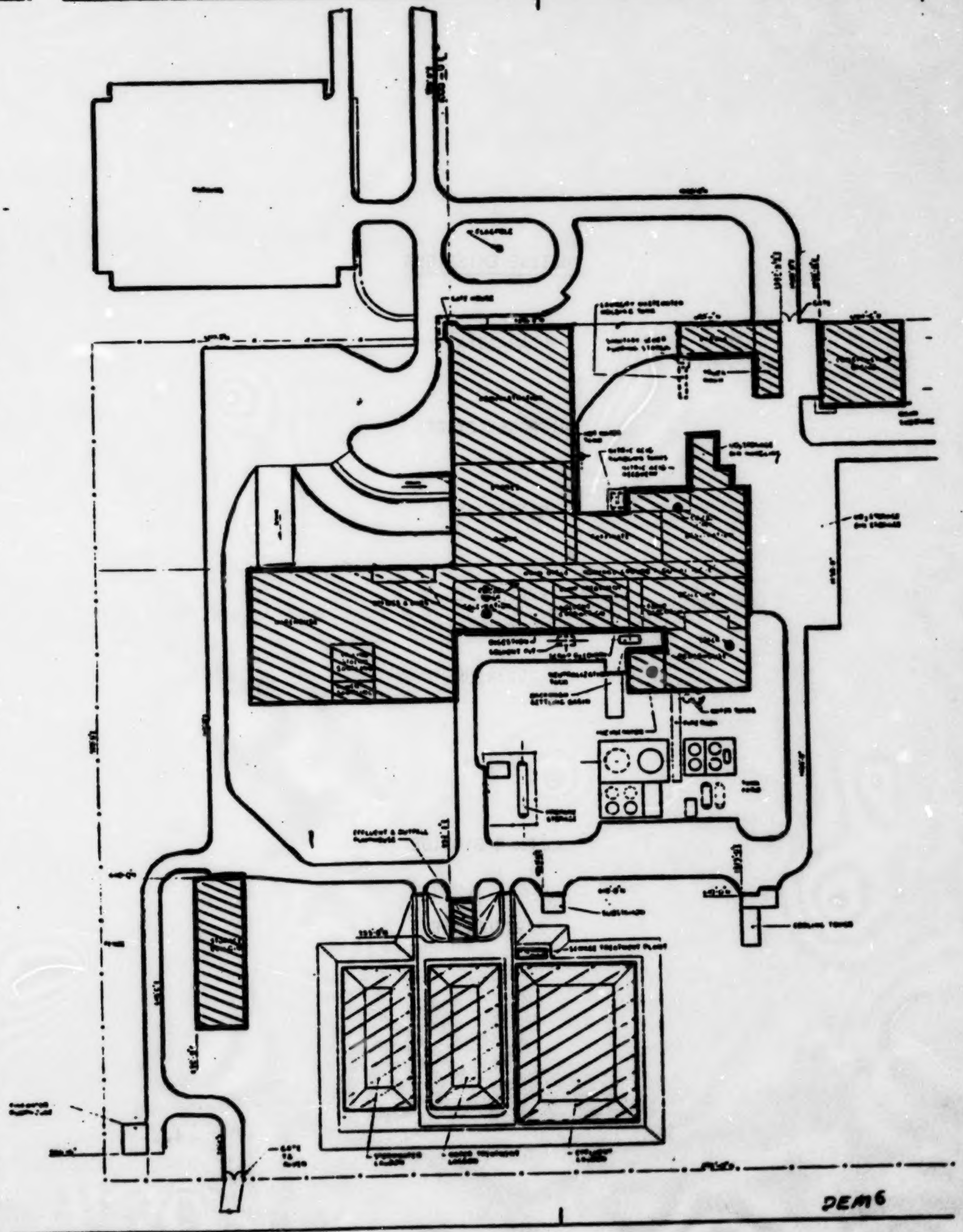
ENVIRONMENT

1. DESIGN
2. OPERATIONAL PLANS
3. RESULTS



17
RAILWAY
ACCESS ROAD
MISSISSAGI
RIVER
E.N.L. PROPERTY BOUNDARY
WOLSTAN POINT
PATRICK POINT
LAKE HURON

DEM 3


EFFLUENT & OUTFALL PUMPHOUSE
SEWAGE TREATMENT PLANT
SUBSTATION
COOLING TOWER

DEM6

ROUTINE EMISSIONS

AMMONIA

NITROGEN OXIDES

NITRIC ACID

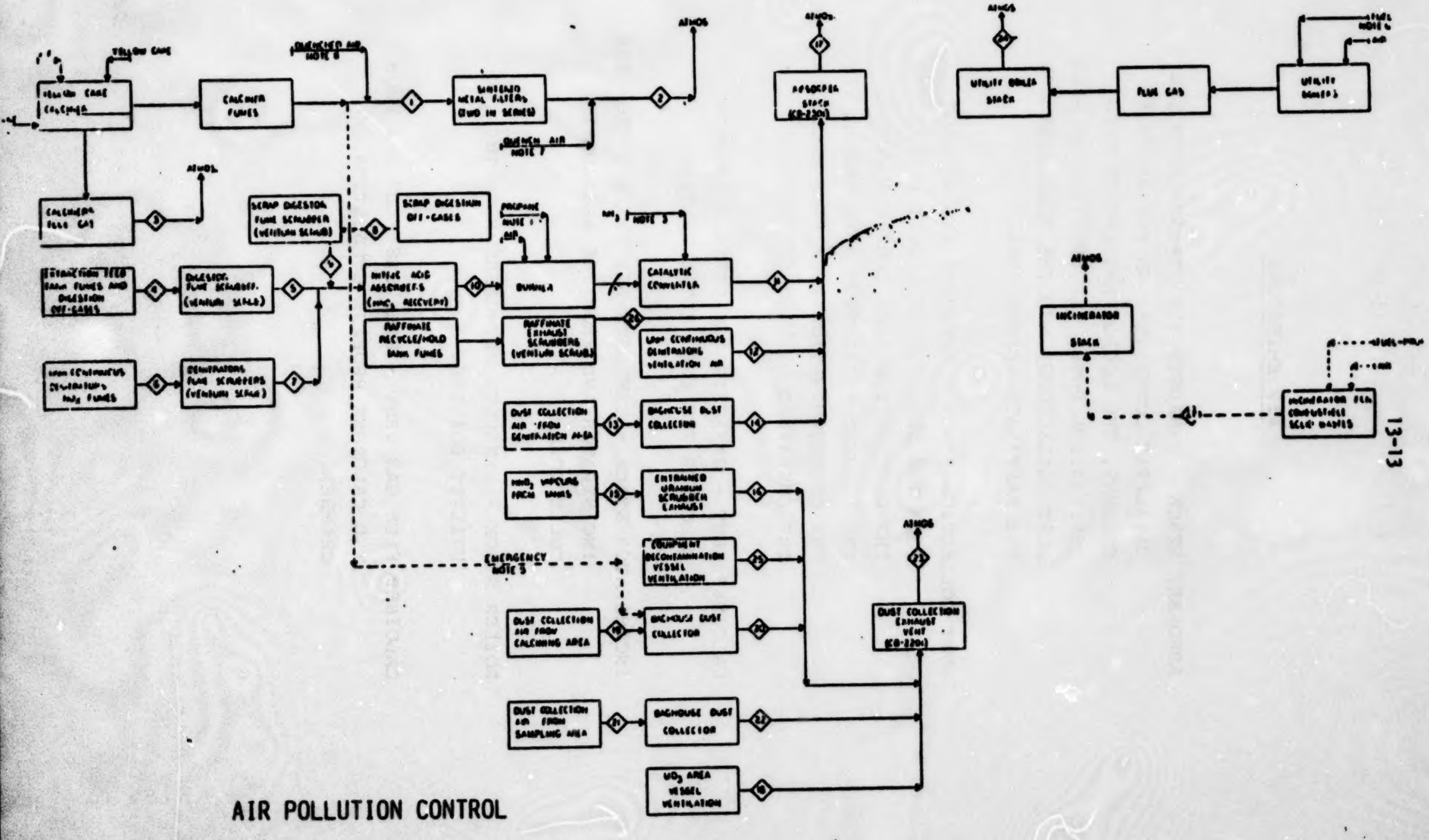
PHOSPHATES

URANIUM

RADIUM

SULPHUR DIOXIDE

PARTICULATE


YELLOW CAKE
YELLOW CAKE CALCINER
CALCINER FUMES
QUENCHED AIR NOTE 8
1
SINTERED METAL FILTERS (TWO IN SERIES)
QUENCH AIR NOTE 7
2
ATMOS
CALCINER FLUE GAS
3
ATMOS.
SCRAP DIGESTION FUME SCRUBBER (VENTURI SCRUB)
8
SCRAP DIGESTION OFF-GASES
9
EXTRACTION FEED TANK FUMES AND DIGESTION OFF-GASES
4
DIGESTER FUME SCRUBBER (VENTURI SCRUB)
5
NITRIC ACID ABSORBERS (HNO_3 RECOVERY)
10
PROPANE NOTE 1
AIR
BURNER
NH_3 NOTE 3
CATALYTIC CONVERTER
11
RAFFINATE RECYCLE/HOLD TANK FUMES
RAFFINATE EXHAUST SCRUBBERS (VENTURI SCRUB)
26
NON-CONTINUOUS DENITRATORS NO_x FUMES
6
DENITRATORS FUME SCRUBBERS (VENTURI SCRUB)
7
NON-CONTINUOUS DENITRATIONS VENTILATION AIR
12
DUST COLLECTION AIR FROM GENERATION AREA
13
BAGHOUSE DUST COLLECTION
14
ABSORBER STACK (CD-2201)
17
ATMOS.
HNO_3 VAPORS FROM TANKS
15
ENTRAINED URANIUM SCRUBBER EXHAUST
16
EMERGENCY NOTE 5
EQUIPMENT DECONTAMINATION VESSEL VENTILATION
25
DUST COLLECTION AIR FROM CALCINING AREA
19
BAGHOUSE DUST COLLECTOR
20
DUST COLLECTION AIR FROM SAMPLING AREA
21
BAGHOUSE DUST COLLECTOR
22
UO_3 AREA VESSEL VENTILATION
18
DUST COLLECTION EXHAUST VENT (CD-2201)
23
ATMOS
UTILITY BOILER STACK
24
ATMOS
FLUE GAS
UTILITY BOILERS
FUEL NOTE 6
AIR
INCINERATOR STACK
ATMOS
INCINERATOR FOR COMBUSTIBLE SOLID WASTES
FUEL-PROP
AIR
27

AIR POLLUTION CONTROL

AIR EMISSIONS

ABSORBER STACK - EXHAUSTS AIR STREAMS FROM THE CATALYTIC CONVERTERS ON THE ABSORBER COLUMNS, THE LOCAL VENTILATION OVER DENITRATION POTS, THE DENITRATION AREA DUST COLLECTORS, AND THE SCRUBBERS FOR THE RAFFINATE VESSEL AREA.

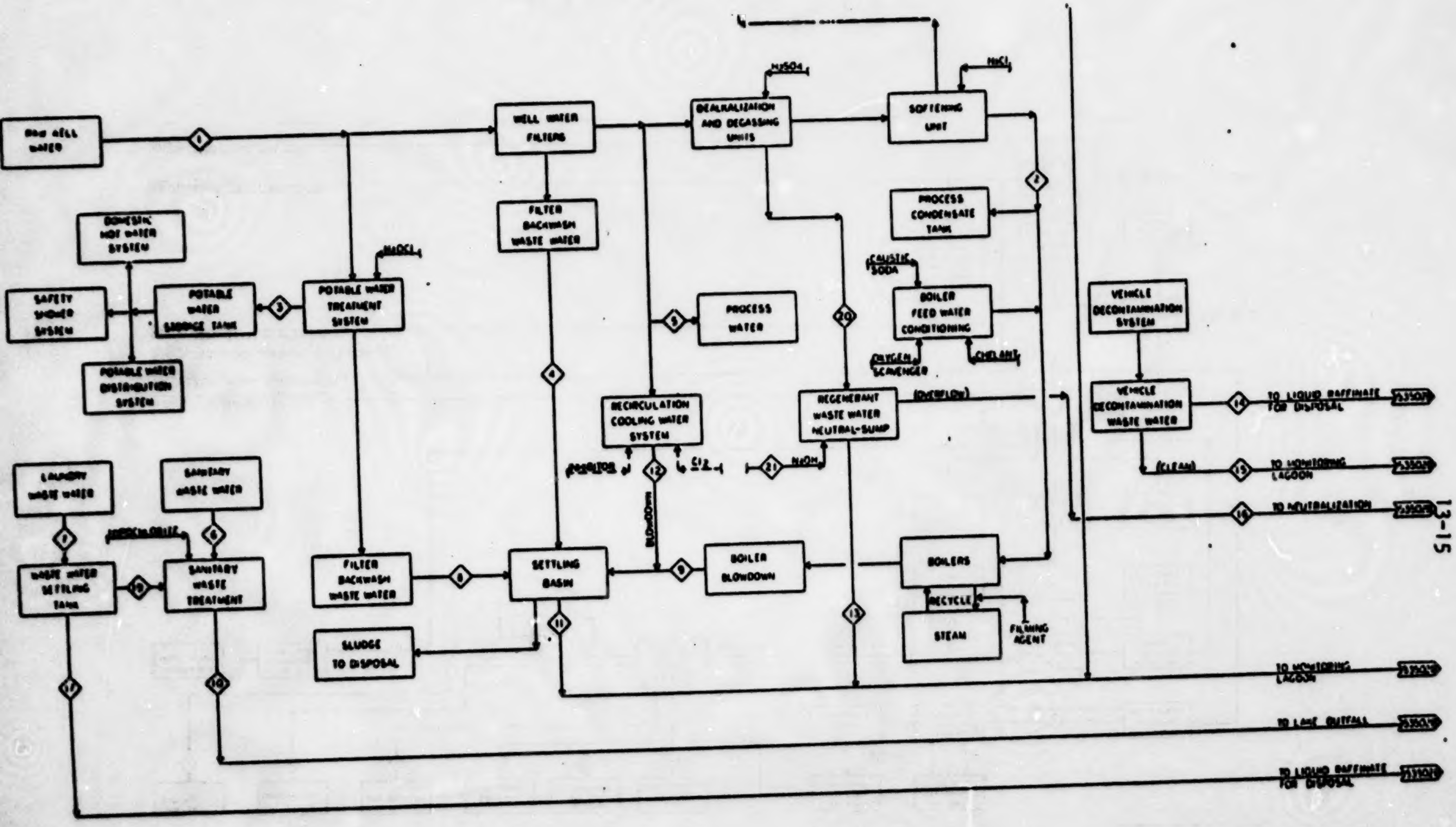
DUST COLLECTION STACK - EXHAUSTS AIR STREAMS FROM THE SAMPLING AREA DUST COLLECTORS, THE CALCINATION AREA DUST COLLECTORS, THE SOLVENT EXTRACTION AREA VESSEL VENTS, THE DECONTAMINATION AREA FUME HOODS, AND THE ENTRAINED URANIUM SCRUBBER.

CALCINER VENT - EXHAUSTS AIR STREAMS FROM THE CALCINER SINTERED METAL FILTERS.

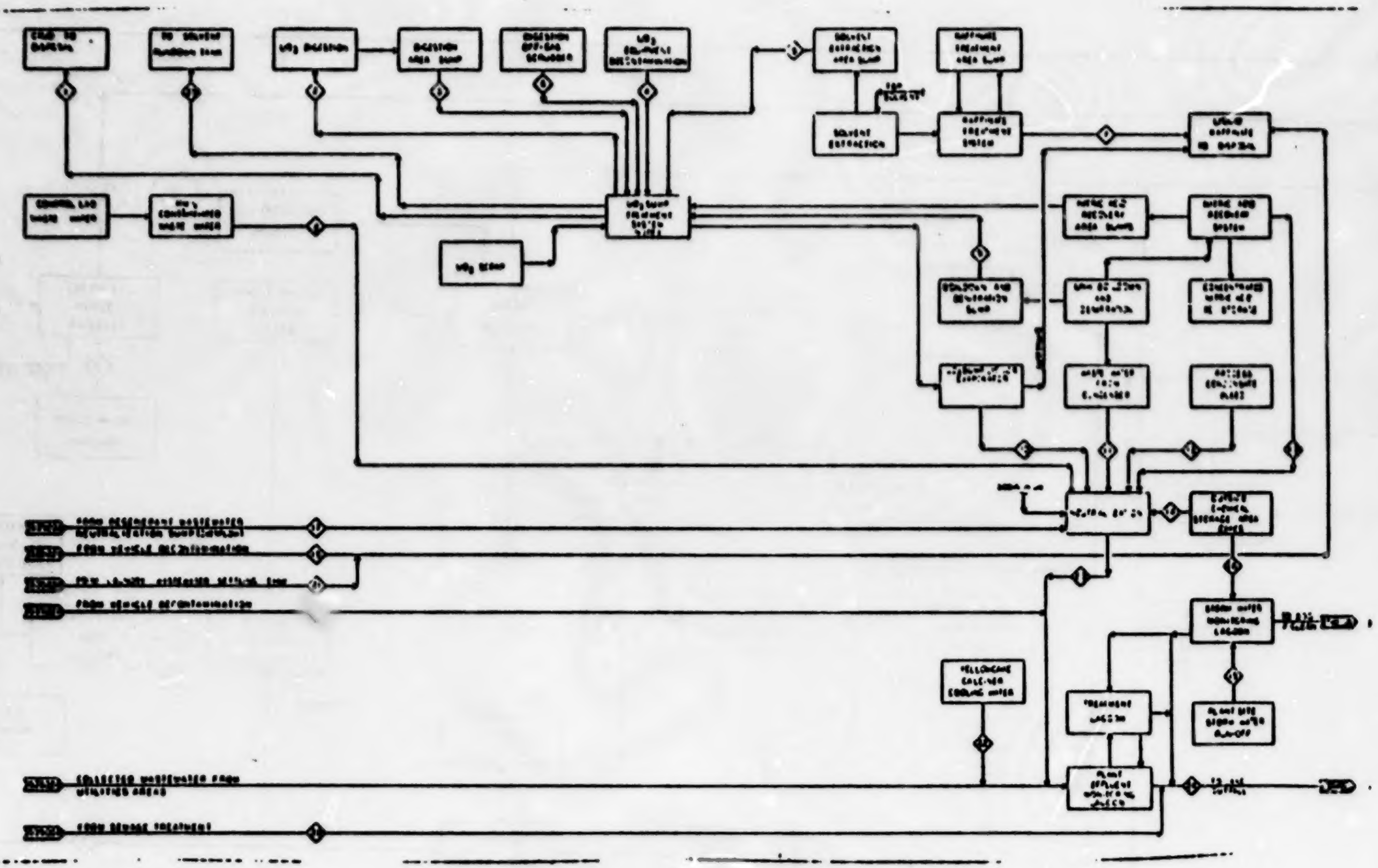
INCINERATOR STACK - EXHAUSTS AIR STREAMS FROM THE INCINERATOR AND FROM THE AREA DUST COLLECTORS.

BOILER STACK - EXHAUSTS OFF-GASES FROM THE TWO UTILITY BOILERS.

CALCINER FLUE GAS VENT - EXHAUSTS FLUE GAS FROM THE CALCINER PROPANE COMBUSTION CHAMBER.


RAW WELL WATER
1
WELL WATER FILTERS
DEALKALIZATION AND DEGASSING UNITS
H2SO4
SOFTENING UNIT
HCl
2
PROCESS CONDENSATE TANK
CAUSTIC SODA
BOILER FEED WATER CONDITIONING
OXYGEN SCAVENGER
CHELANT
DOMESTIC HOT WATER SYSTEM
SAFETY SHOWER SYSTEM
POTABLE WATER STORAGE TANK
3
POTABLE WATER TREATMENT SYSTEM
NaOCl
POTABLE WATER DISTRIBUTION SYSTEM
FILTER BACKWASH WASTE WATER
4
5
PROCESS WATER
20
RECIRCULATION COOLING WATER SYSTEM
INHIBITOR
12
Cl2
BLOWDOWN
21
NaOH
REGENERANT WASTE WATER NEUTRAL-SUMP
(OVERFLOW)
VEHICLE DECONTAMINATION SYSTEM
VEHICLE DECONTAMINATION WASTE WATER
14
TO LIQUID RAFFINATE FOR DISPOSAL
(CLEAN)
15
TO MONITORING LAGOON
16
TO NEUTRALIZATION
LAUNDRY WASTE WATER
SANITARY WASTE WATER
7
6
HYPOCHLORITE
WASTE WATER SETTLING TANK
19
SANITARY WASTE TREATMENT
FILTER BACKWASH WASTE WATER
8
SETTLING BASIN
9
BOILER BLOWDOWN
BOILERS
RECYCLE
STEAM
FILMING AGENT
11
13
SLUDGE TO DISPOSAL
10
17
TO MONITORING LAGOON
TO LAKE OUTFALL
TO LIQUID RAFFINATE FOR DISPOSAL

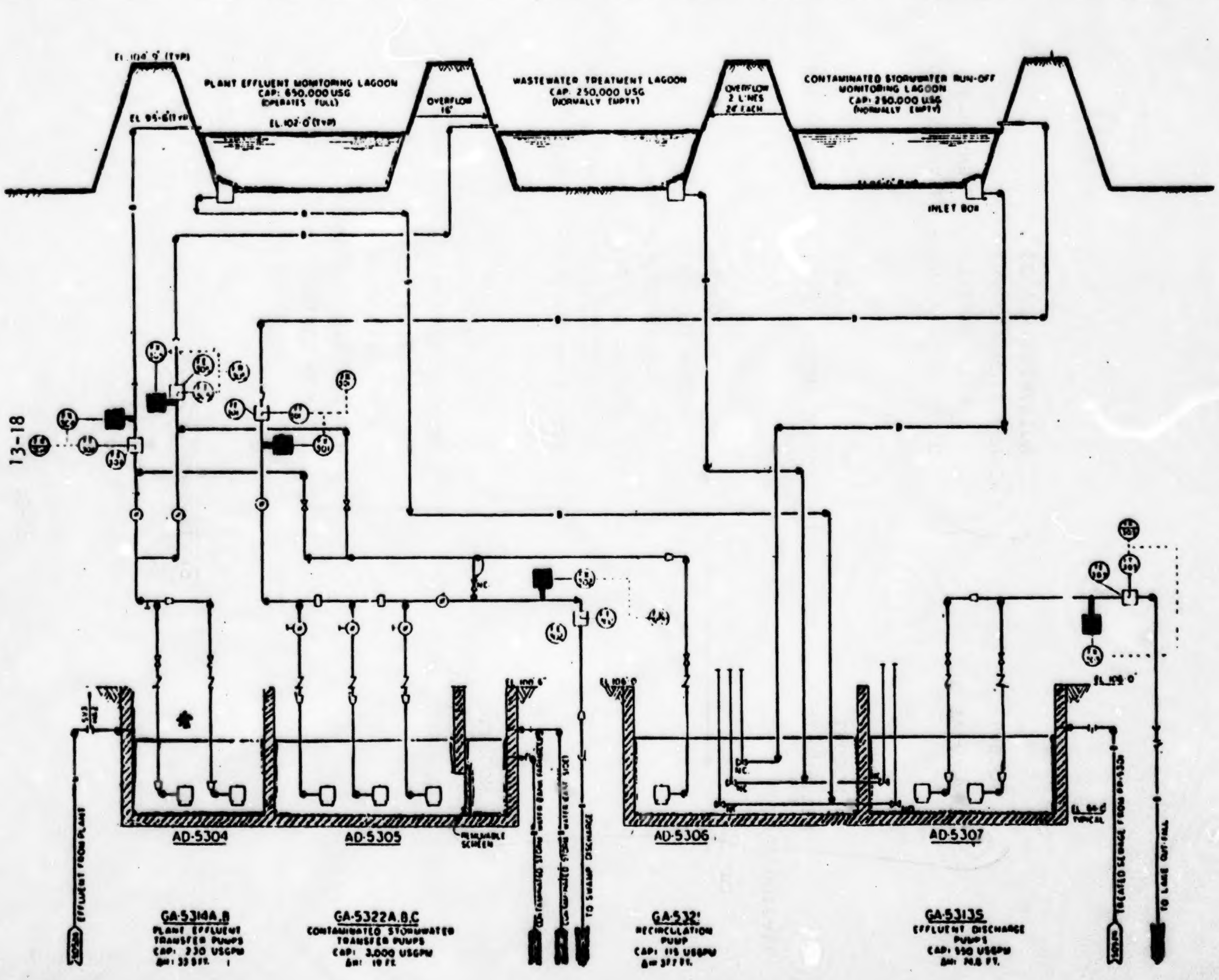
UTILITIES WASTEWATER TREATMENT



PROCESS WASTEWATER TREATMENT

WASTEWATER STREAMS

PROCESS	- CALCINER C.W.	TO MONITORING LAGOON	TO OUTFALL
	- CONCENTRATOR O.H.		
	- SUMP EVAP. O.H.		
	- LAB WATER		
	- CONDENSATE BLEED		
	- TRUCK WASH		
UTILITIES	- FILTER BACK WASH		
	- BOILER BLOWDOWN		
	- COOLING WATER B.D.		
	- REGENERANT W.W.		
	- LAUNDRY WATER - SEWAGE PLANT WW	→	
STORMWATER	- FIRST FLUSH DESIG. AREAS	TO STORMWATER LAGOON *	
CONTAMINATED STREAMS	- ANY OF THE ABOVE STREAMS WITH UNACCEPTABLE LEVELS OF CONTAMINATION	TO TREATMENT LAGOON	


13-18
EL 104'9" (TYP)
PLANT EFFLUENT MONITORING LAGOON
CAP: 650,000 USG
(OPERATES FULL)
EL 95-6 (TYP)
EL 102'0" (TYP)
OVERFLOW
16"
WASTEWATER TREATMENT LAGOON
CAP: 250,000 USG
(NORMALLY EMPTY)
OVERFLOW
2 LINES
24" EACH
CONTAMINATED STORMWATER RUN-OFF
MONITORING LAGOON
CAP: 250,000 USG
(NORMALLY EMPTY)
INLET BOX
EL 109'0"
EL 84'0"
TYPICAL
REMOVABLE
SCREEN
AD-5304
AD-5305
AD-5306
AD-5307
EFFLUENT FROM PLANT
CONTAMINATED STORM WATER (TANK FARM)
CONTAMINATED STORM WATER (EAST SIDE)
TO SWAMP DISCHARGE
TREATED SEWAGE FROM PP-5301
TO LAKE OUT-FALL
GA-5314A,B
PLANT EFFLUENT
TRANSFER PUMPS
CAP: 230 USGPM
ΔH: 33 FT.
GA-5322A,B,C
CONTAMINATED STORMWATER
TRANSFER PUMPS
CAP: 3,000 USGPM
ΔH: 10 FT.
GA-5321
RECIRCULATION
PUMP
CAP: 115 USGPM
ΔH: 37 FT.
GA-5313S
EFFLUENT DISCHARGE
PUMPS
CAP: 550 USGPM
ΔH: 76.8 FT.

2. OPERATIONAL PLANS

PRE-OPERATIONAL MONITORING

THE CONTROL PROCESS

EMISSIONS MONITORING

AMBIENT MONITORING

WASTE MANAGEMENT

BLIND RIVER AND AREA ENVIRONMENTAL MONITORING COMMITTEE (BRAEMC)

PARTICIPATIVE MANAGEMENT

3. RESULTS

AIR - AUGER SAMPLING

PROGRAM OBJECTIVES

1. BASELINE LEVELS OF ENVIRONMENTAL CHEMISTRY
2. PHYSICAL CHARACTERISTICS OF THE ENVIRONMENT
3. ESTIMATE VARIATIONS OF 1 AND 2 WITH TIME AND LOCATION

WHY??

1. CHECK ENVIRONMENTAL MODELS
2. CLEARLY IDENTIFY SIGNIFICANT CHANGES DUE TO REFINERY OPERATION
3. OPERATIONAL MONITORING PROGRAM DESIGN
4. IDENTIFY TECHNICAL PROBLEMS

SUMMARY OF RESULTS

PARAMETERS WHICH WERE AS EXPECTED (OTHER STUDIES, LITERATURE)

- AIR CHEMISTRY
- WIND SPEED, DIRECTION
- NOISE LEVELS
- METAL, RADIONUCLIDES IN VEGETATION
- MAJOR ELEMENTS, RADIONUCLIDES IN VEGETATION
- MAJOR ELEMENTS AND TRACE ELEMENTS IN SURFACE WATERS
- GROUNDWATER FLOWS IN SW DIRECTION
- GROUNDWATER CHEMISTRY VARIES WITH SPACE AND TIME
- SEDIMENT CHEMISTRY
- SEDIMENTS ARE COARSE AND SANDY

SPARSE ZOOBENTHIC POPULATION
SPARSE CLAM COMMUNITY

- LEVELS OF METALS AND RADIONUCLIDES IN FISH

UNEXPECTED PARAMETERS

- MOLYBDENUM LEVELS IN SOIL
- AMMONIA, NITRATE, PHOSPHOROUS, & SILVER IN SURFACE WATERS
- SILVER IN SEDIMENTS
- AL, AG, B, CR, CU, FE, MN, NI, TI, V IN FISH

MAJOR CONCLUSIONS

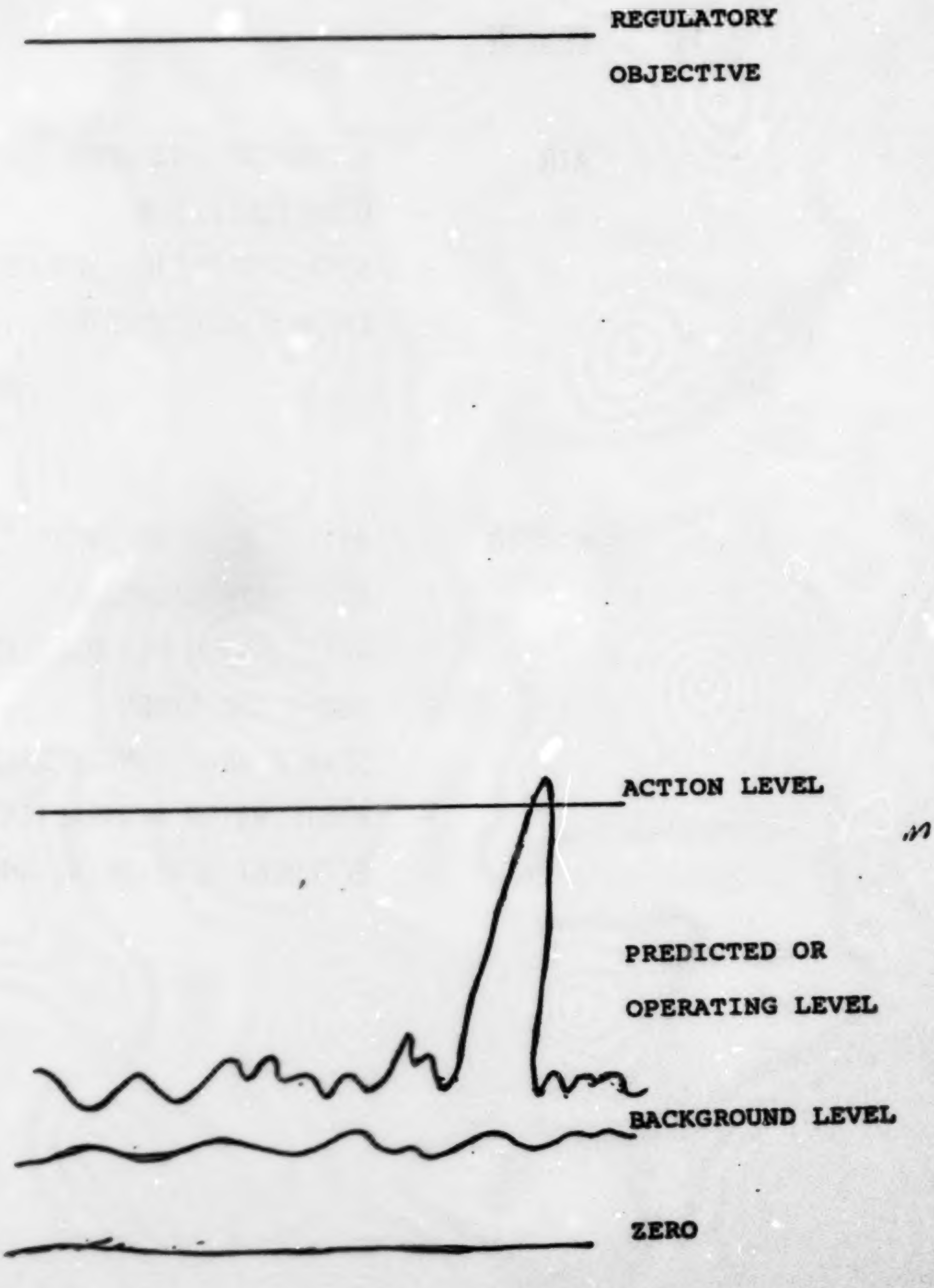
1. HAVE ESTABLISHED BASELINE LEVELS FOR PHYSICAL AND CHEMICAL CHARACTERISTICS OF ENVIRONMENT

2. WILL BE ABLE TO DETECT SIGNIFICANT CHANGES IN ENVIRONMENT DUE TO THE REFINERY IF THEY OCCUR

3. QUESTIONS OUTSTANDING

   - SILVER IN WATER, SEDIMENTS?
   - PHOSPHOROUS LEVELS DO NOT CORRELATE WITH CHLOROPHYLL-A ?
   - EXTERNAL GAMMA LEVELS AT HI END OF CANADIAN RANGE ?
   - DETECTION LIMITS FOR SOME ELEMENTS TOO HI ?

THE CONTROL PROCESS


REGULATORY OBJECTIVE
ACTION LEVEL
PREDICTED OR OPERATING LEVEL
BACKGROUND LEVEL
ZERO

1. DESIGN

AIR
- SCRUBBERS AND VENTS
- DUST COLLECTORS
- SINTERED METAL FILTERS
- CATALYTIC CONVERTERS - NO_x

LIQUID
- NITRIC ACID RECOVERY
- RAFFINATE DISPOSAL
- WASTE WATER NEUTRALIZATION
- SUMPS AND DYKES
- SEWAGE AND LAUNDRY WATER TREATMENT
- WASTE WATER NEUTRALIZATION
- EFFLUENT MONITORING AND TREATMENT

EMISSION MONITORING OBJECTIVES

DETERMINING CONCENTRATION OF VARIOUS CONTAMINANTS IN AIR AND WATER AT THEIR POINT OF RELEASE TO THE ENVIRONMENT TO:

1. EVALUATE THE PERFORMANCE OF EMISSION CONTROL EQUIPMENT
2. DETECT TRENDS IN CONCENTRATIONS OF EMITTED CONSTITUENTS WHICH MAY INDICATE A NEED FOR CORRECTIVE ACTION
3. DETECT THE NATURE AND EXTENT OF UNPLANNED RELEASES
4. EVALUATE COMPLIANCE WITH THE APPROPRIATE REGULATORY STANDARDS AND OBJECTIVES

DEM

EMISSION MONITORING

AIR - 6 POINTS OF DISCHARGE

- STACK SAMPLING

- CONTINUOUS ANALYZER

WATER - POINT OF DISCHARGE

- UPSET AND COMPLIANCE MONITORING

- STORMWATER

AMBIENT MONITORING

AIR - SUSPENDED PARTICULATE

- DUSTFALL DEPOSITION RATE
- GAMMA RADIATION
- METEOROLOGICAL MONITORING

SULPHATION RATES, RADON,
NO_x LEVELS, NOISE,
VEGETATION, SOILS.

WATER - N. CHANNEL

- GROUNDWATER

RIVER, SEDIMENT, ZOOBENTHOS, FISH

WASTE MANAGEMENT

- SEGREGATION
- OFF-SITE DISPOSAL
- INCINERATION AND DISPOSAL
- ON-SITE STORAGE IF CONTAMINATED
- DRUMS -- RECYCLED/SCRAP DISPOSAL
- METAL -- DECONTAMINATED AND SCRAP DISPOSAL

BRAEMC

- FORMAL LINK BETWEEN PUBLIC AND BRR
- MEETS MONTHLY/QUARTERLY
- PLANT AMBIENT AND EMISSIONS PERFORMANCE REVIEW

PARTICIPATIVE MANAGEMENT

Environment Committee

- CROSS SECTION OF ALL BRR EMPLOYEES
- REVIEWS AND MONITORS DATA, REPORTS, PROGRAMS, PROCEDURES
- ASSESSES PROCEDURES, NEW EQUIPMENT, ETC. FOR ENVIRONMENTAL IMPACT
- RECOMMENDS FOR MANAGEMENT APPROVAL
- REPRESENTS BRR TO REGULATORY AGENCIES, BRAEMC, AND PUBLIC

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